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DEPARTMENT OF THE INTERIOR

UNITED STATES BUREAU OF MINES
JOHN W. FINCH, DIRECTOR

REPORT OF INVESTIGATIONS

PERMISSIBLE ELECTRICALLY OPERATED
ROCK-DUST DISTRIBUTORS



BY

L. C. ILSLEY, E. J. GLEIM, AND H. B. BRUNOT

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REPORT OF INVESTIGATIONS
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PERMISSIBLE ELECTRICALLY OPERATED ROCK-DUST
DISTRIBUTORS^{1/}

By L. C. Ilsley^{2/}, E. J. Gleim^{3/}, and H. E. Brunot^{4/}

INTRODUCTION

The general nature of the special constructional features that are required to provide safety from explosion hazards on machines approved as permissible by the Bureau of Mines is fairly well known to mining men who are at all interested in safety. These requirements are outlined in the schedules, under the provisions of which the investigations of such machines are conducted. Also, the method of conducting inspections and tests of such machines has been described in Bureau of Mines Bulletin 305, Inspection and Testing of Mine Type Electrical Equipment for Permissibility, published in 1929. A more detailed knowledge of the special safety features of a particular machine is often useful and sometimes essential for inspectors, safety engineers, maintenance men, and engineers in selecting machines for a particular application. Three bulletins that give information of this nature have been published by the bureau, viz.,

Bulletin 313, Permissible Storage-Battery Locomotives
and Power Trucks.

Bulletin 343, Permissible Coal-Handling Equipment.

Bulletin 382, Permissible Coal-Cutting Equipment.

Report of Investigations 3309, Permissible Electrically Operated Air Compressors, which contains similar information, was issued in August 1936, and a Report of Investigations 3326, Permissible Electrically Operated Room Hoists, was issued in January 1937. Thus, the present paper is the third to give detailed information concerning permissible machines of types not included in bulletins, and covers all the rock-dust distributors approved prior to October 1, 1936.

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Nine rock-dust distributors have received the official approval of the Bureau. Most of them are arranged for connection to a permissible storage-battery power truck, which not only supplies the energy for operating the blower and the agitator of the distributor but also furnishes motive power for transporting the machine about the mine. This applies only to direct-current machines for voltages of 250 or less, since the battery voltages of approved power trucks do not exceed 250 volts. For 500-volt direct-current and for alternating-current machines a safe means of haulage, such as an approved battery locomotive or animal haulage, should be used. Both high- and low-pressure types of machines are included in the approved rock-distributing machines. Three manufacturers build the nine machines that have been approved.

DETAILED DESCRIPTIONS

Mine Safety Appliances Co.

First Approved Rock-Dust Distributor

Approval No. 130

The Mine Safety Appliances Co. rock-dust distributor was approved on November 5, 1926. The machine, as approved, is arranged for operation by power from a permissible storage-battery power truck. Approval No. 130, which was the first approval issued for a rock-dust distributor, was assigned to this outfit.

The motor and fan are mounted on the rear of the four-wheel truck; the controller, headlight, and discharge nozzle are at the front. A covered hopper for the rock dust occupies the center. The nozzle is controlled by the operator, who directs the discharge of rock dust as desired.

Current for operating the machine is received through a two-conductor rubber-clad cable that enters the controller through an asbestos-packed stuffing box. The control is a push-button-operated, automatic, magnetic starter of the counter E.M.F. acceleration type, having one step of starting resistance. It is connected for low-voltage protection. All the parts are carried by a panel of insulating material and consist of a two-pole line contactor, a single-pole accelerating contactor, a magnetic overload relay, the starting resistance, the push-button station, and two fuses for the headlight circuit. The positive wire of the supply cable is connected to the positive terminal of the control, i.e., the terminal that leads to the overload relay. The push buttons are operated by plungers that go through the top of the explosion-proof enclosing case.

Connections are made between the motor and the control by means of a three-conductor, rubber-clad cable that enters each of the compartments through an asbestos-packed stuffing box. Connections to the headlight are made by a two-conductor, No. 18, rubber-clad cord; the lead entrances at the

headlight and controller are asbestos-packed stuffing boxes.

All the cables are protected by rubber-hose conduit. In addition, the cable to the motor and the cable from the power supply are protected by rigid metal conduits along the side of the hopper. The hose conduit on the latter cable extends back to the plug, which makes connection to a permissible storage-battery power truck. The ends of the hose conduits are slipped over extensions on the stuffing-box nuts and are secured by hose clamps.

The motor was a Westinghouse type 53-SK compound-wound motor, rated 5 horsepower for continuous duty with a maximum temperature rise of 55° C. The motor frame is a cylinder of rolled-steel plate bored out to take the pole pieces and finished at the ends for the joints with the cast-iron bearing brackets that close its ends and carry the ball bearings for the shaft. The joints are of the rabbet type; a machined lip on the rim of the bracket fits snugly into the bore of the frame, while a shoulder on the rim is held against the end of the frame by eight 1/2-inch tap bolts; a right-angle joint is thus formed with wide metal surfaces in contact between the parts. The bolts go into tapped bottomed holes in the frame and are secured by lock washers. Inspection of the commutator and brushes is allowed by four equally spaced circular holes through the front bracket with their axes perpendicular to that of the armature. A screw cover fits into each of the inspection openings, and the four covers are sealed in place to prevent removal by unauthorized persons. The ball bearings fit bored holes through the centers of the bearing brackets and each is held in place by two bearing caps, one inside and one outside the motor. Four studs welded to the inside cap pass through holes in the bearing bracket and the outer bearing cap to hold the parts in place. The nuts are secured by lock washers. The studs are 3/8-inch for the commutator-end bearing assembly and 1/2-inch for the pinion-end assembly. The bearing caps make right-angle joints with the bearing brackets, with wide metal surfaces in contact between the parts. To prevent the discharge of flame along the shaft, the latter has a close running fit through the inside bearing-caps for a considerable distance. The outside bearing-caps give additional protection. The grease holes should be closed to obtain full benefit from the outside caps. The wires enter the side of the commutator-end bearing bracket in the form of a three-conductor, rubber-clad cable through an asbestos-packed stuffing-box. The stuffing-box nut is extended to slip inside the hose conduit that protects the cable. A hose clamp secures the hose in place on the nut.

The control compartment consists of three main parts - a cast-iron base, a dome-shaped cast-iron cover, and a brass clamping ring or nut. The joint between the cover and base is of the rabbet type, with wide metal surfaces in contact. The clamping ring screws on the base and holds the cover down in place. It is sealed to prevent tampering. The metal surfaces in contact between the clamping ring and cover give additional protection against the discharge of flames. The wires enter and leave the compartment as rubber-clad cables and cord through asbestos-packed stuffing boxes in the side of the base. The cables are protected by hose conduits attached in the same manner as that at the motor. The long, close fits of the push-button rods through the top of the cover prevent the discharge of flames at these points.

The headlight was manufactured by the Mancha Storage Battery Locomotive Co. It consists of a cylindrical brass shell with the front and rear covers screwed on its ends. The threaded joints make effective seals to prevent the discharge of flames. The covers are locked by a rod that passes through lugs cast on the case and is secured by a padlock. The ends of the rod rest in notches in the peripheries of the covers. The bull's-eye lens seats against a spherical surface machined in the front cover and is held by a ring that screws against it from the rear; a set screw prevents the ring from loosening. The wide surfaces in contact between the glass, the front cover, and the retaining ring have proved effective in preventing the discharge of flames. The lamp socket is held by a spider designed to slide inside the cylindrical shell. The reflector is clamped between the spider and a projecting ring within the shell. The lead-entrance stuffing box is in the rear cover.

Diamond Machine Co.

"Legrabon" Rock-Dust Distributor

Approval No. 131

The Diamond Machine Co. Legrabon rock-dust distributor was approved on December 28, 1926. The machine as approved is arranged for operation by power from a permissible storage-battery power truck.

Figure 1 shows the Diamond Machine Co. Legrabon rock-dust distributor as first approved. An extension of the approval has allowed the addition of a platform for the operator to stand on when feeding the hopper, and a platform over the motor to aid in handling the bags of rock dust from the cars to the hopper. An optional arrangement is permitted in which cable a is taken around the back and over the top of switch-and-fuse box b. Connection plug c connects two-conductor, rubber-clad cable a which is protected by a hose conduit throughout to a permissible storage-battery power truck. The two conductors enter fuse-and-switch box b through two asbestos-packed stuffing boxes under a sheet-metal shield on its far side. A clamp holds the end of the hose, and two wood clamps space and hold the two conductors where they are separated between the end of cable and the lead entrances. Connections between the switch-and-fuse box and the motor and control unit are made by two single-conductor, rubber-clad cables under sheet-metal guard d. These cables leave the switch-and-fuse through two asbestos-packed stuffing boxes, but connections to the motor unit are through insulated studs.

The switch-and-fuse compartment b contains a double-pole switch and two 100 ampere fuses, one for each side of the circuit. The switch gives one break per pole. The contacts are surrounded by barriers of asbestos board to form arcing chambers and permanent magnets are used to furnish magnetic blow-outs for arcs formed when the switch is opened under load. The explosion-proof case enclosing the switch and fuse is a strong cast-iron box with a circular opening through the front closed by a ribbed aluminum cover, which screws into the opening. Bar e, which connects the switch levers, prevents unscrewing the

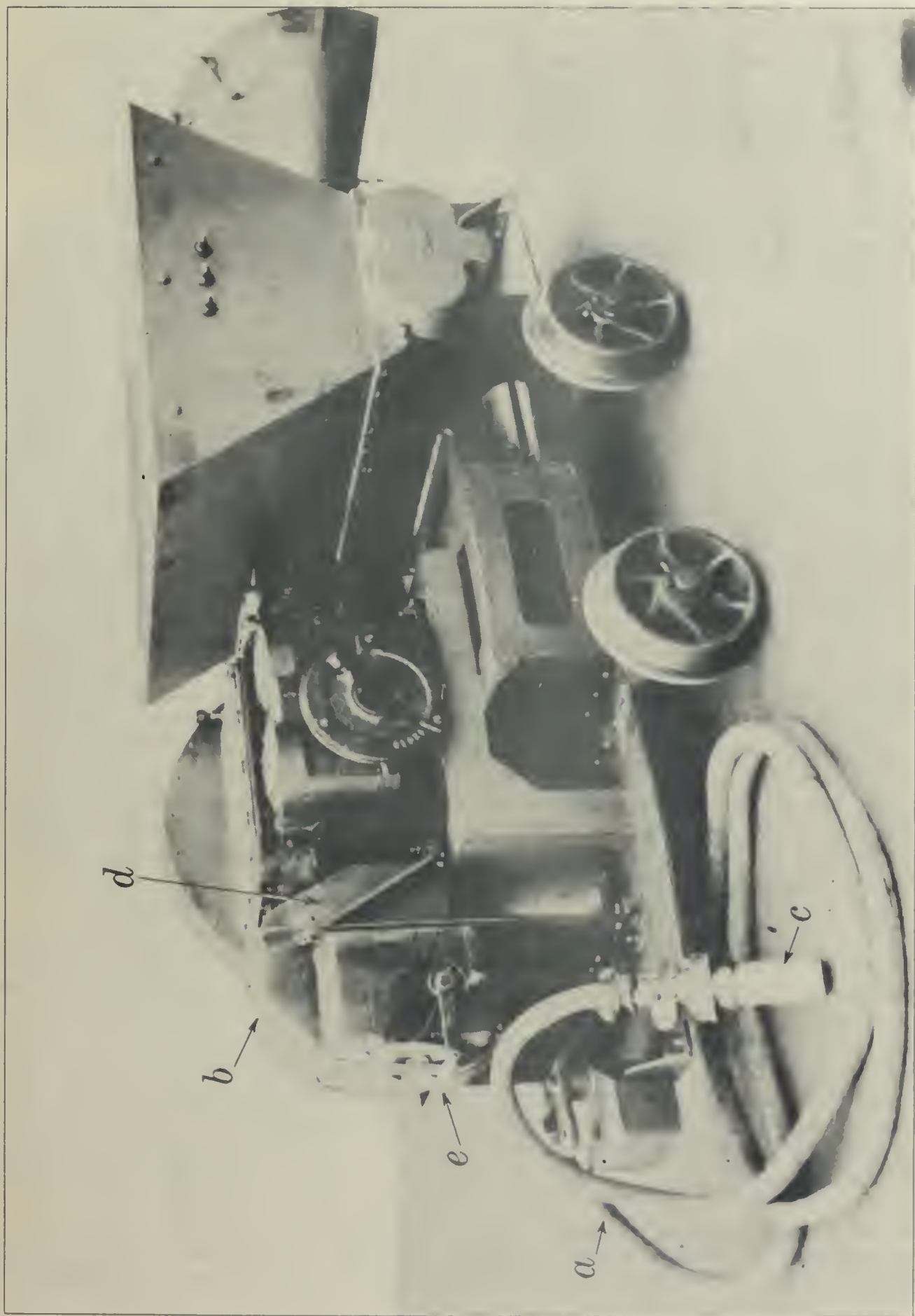



Figure 1.- Diamond Machine Co. Legrabon rock-dust distributor.



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cover. Downward movement of the bar or handle opens the switch and allows the cover to be unscrewed. An arm hinged to the case and attached by a pivot to the center of the cover prevents the switch from being closed after the cover has been unscrewed. The interlocking arrangement permits renewal of fuses without hazard. All the wires enter the compartment through asbestos-packed stuffing boxes. The screw threads of the cover and long, close-fitting bearings for switch-operating shafts prevent the discharge of flame from the case.

The MM-146-A motor is rated at 16 horsepower. The motor and resistor are enclosed in the same steel casting but are separated by a partition; the resistor compartment is at the top. The starting box is fastened to the side of the motor frame, which is machined to form an explosion-proof joint with the box and thus close its one open side. The motor shell is cast closed, except for a circular bored hole in each end, two inspection holes, one at each side of the center just above the commutator, and an opening for the lead entrance. The bored hole at the commutator end is large enough to allow the armature to be inserted and is closed by the cast-iron commutator-end bearing housing. The hole at the gear end is much smaller and is closed by the cast-iron gear-end bearing housing. The inspection holes above the commutator are threaded for screw-type hand-hole covers. The commutator-end bearing housing is fastened in place by three 1/2 inch studs with nuts and lock washers; the studs are riveted inside the motor frame. The gear-end housing is fastened by three 1/2 inch studs with nuts and lock washers; the studs are riveted to the bearing housing. The shaft fits closely through baffles on the inner ends of the bearing housings. At the commutator end a sleeve on the shaft provides the close fit necessary to prevent discharge of flame along the shaft; the ball bearing and the running fit of a special washer on the end of the shaft in the bearing cap give some additional protection, but drain holes through the cap prevent it from completely closing the entrance to the bearing. At the pinion end the ball bearing and the running fit between the bearing nut and the bearing retainer give additional protection against discharge of flame. The grease cups should be in place, as the grease holes lead to the inner side of the ball bearings. The brush rigging is fastened by two screws through the top of the motor frame; these and the two inspection covers should be sealed.

The resistance compartment is closed by a 3/8 inch brass cover held by thirteen 1/2 inch cap screws secured by lock washers; the screw holes are bottomed in the casting. The class 21 starting box is fastened by eight 1/2 inch cap screws in bottomed holes in the magnet-frame casting; these are secured by lock washers. The starting box is a two-part flanged casting, the parts of which are bolted together by eight 1/2 inch bolts with nuts secured by lock washers. The cover has been changed to a heavy, malleable-iron casting by extension of approval instead of the ribbed, cast-iron cover shown by the figure. The wide metal surfaces in contact at the joints and the long, close, bearing fits prevent discharge of flame from the resistance and starting-box compartments.

The leads between compartments go through the partitions in asbestos-packed stuffing boxes. The incoming circuit enters the motor compartment by two insulated studs carried by a 1/2 inch steel plate bolted over an opening

through the magnet frame by four 1/2 inch cap screws secured by lock washers; the screw holes are bottomed. The wide metal surfaces in contact at the joint between the plate and the frame prevent discharge of flame. A terminal cover protects the outside connections.

Mine Safety Appliances Co.

Type B Rock-Dust Distributor

Approval No. 137

The Mine Safety Appliances Co. type B, high-pressure, rock-dust distributor was approved July 2, 1927.

The MI-146 motor unit, which is used on the Mine Safety Appliances Co. type B high-pressure rock-dust distributor, is identical with the motor unit described for Approval No. 131. The switch and fuse box is also the same as for Approval No. 131, except that two headlight fuses and a stuffing-box lead entrance for the headlight cable have been added. The headlight is the same as that described for Approval No. 130. The wiring between the switch-and-fuse, the motor, and the headlight is protected by rubber-hose conduit. The machine is arranged for operation by a permissible storage-battery power truck. The power intake line is a 25-foot length of two-conductor, rubber-clad, No. 3, parallel duplex cable. This is spliced to two single-conductor, No. 4, rubber-clad cables for the individual stuffing-box lead entrances to the switch-and-fuse, and is fitted with a standard plug connection at the other end for the power truck. The cable is protected by a rubber-hose conduit between the machine and the power truck.

Extension of Approval

The only change that has been allowed in the construction features is the replacing of the ribbed, cast-iron cover of the starting box by a heavy, malleable-iron cover, as was done in connection with the starting box of the machine covered by Approval No. 131.

Mine Safety Appliances Co.

Types 125, 150, and 350 Rock-Dust Distributors

Approvals Nos. 146 and 146A

The Mine Safety Appliances Co. type 125 rock-dust distributor was approved for 250-volt service January 20, 1928, and for 500-volt service on April 3, 1928. The type 150 machine was covered by extension of the approval dated June 12, 1928. The 250-volt type 350 machine was allowed by extension of approval dated February 26, 1929, and the 500-volt machine by extension of approval dated August 13, 1934.

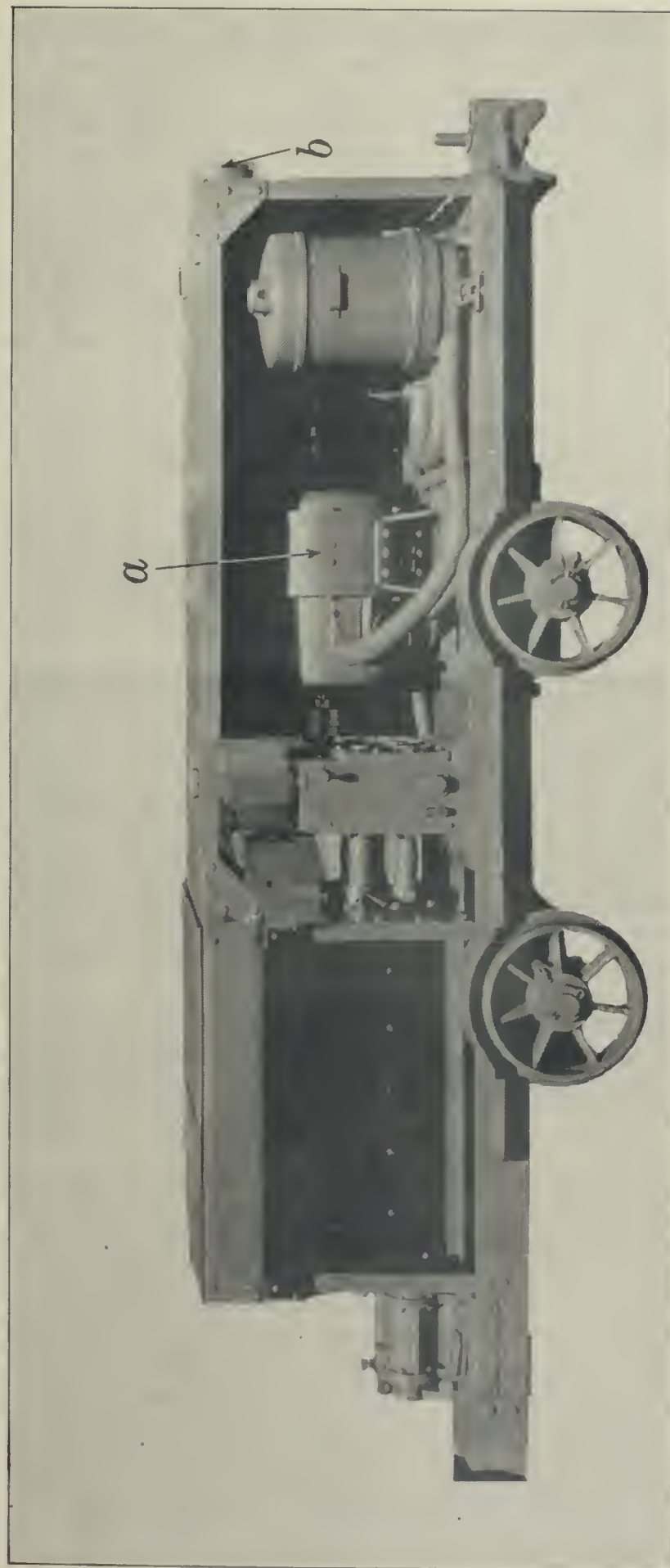


Figure 2.- Mine Safety Appliance Co. type 150 rock-dust distributor.

Figure 2 shows the No. 150 machine. The type 125 machine is similar to the type 150 machine, the chief differences between the two being the arrangement of the motor, blower, and auxiliary drive. On the No. 125 machine motor a is turned 180° from the position shown in figure 2. This places the blower next to the end of the hopper instead of beside the control as the blower is directly coupled to the motor shaft. The auxiliary drive for the agitators and feed screws is through a chain from the motor-shaft extension to the blower on the No. 125 machine, instead of by a second motor-shaft extension as on the No. 150 machine. The type 350 machine differs from the other two; it has a fan-type blower, a stationary nozzle, and a lower hopper. The motor is set crosswise on the truck. The auxiliary drive is by a chain from one shaft extension, and the blower fan is on the other shaft extension. On 500-volt No. 350 machines, a different motor and control are used from those for 250-volt machines. Also, a headlight resistance compartment is added. Further, the headlight and discharge nozzle are on the hopper end of the machine and the trailing cable leaves from the motor and control end of the machine.

Machines for 250-volt service carry one Mancha headlight; for 500-volt service two Westinghouse type S headlights, connected in series, are used, except that the 500-volt No. 350 machine uses one Mancha headlight and a headlight resistance.

The 250-volt machines are arranged to receive power from a permissible storage-battery power truck through a short length of 2-conductor, rubber-clad cable protected by hose conduit. A connection plug such as that shown at c, figure 1, is used for the connection at the power truck. The 500-volt machines have a fused trolley tap and a rail clamp or other suitable means for connecting to the power-supply circuit. All of the wiring on the machines is of rubber-clad cable and is protected by hose conduits. Metal conduits are used in addition to the hose where the wiring is exposed along the sides of the hoppers. The power intake cables are clamped to prevent strain on the connections at the controller and connection plug.

None of the rock-dust distributors is equipped with self-propelling mechanism. Transportation of 500-volt machines is restricted to animal haulage; the 250-volt machines are moved by the battery power truck, which furnishes the power for their operation.

Westinghouse Electric & Manufacturing Co. type 103-SK motors, rated at 12 horsepower for 24 hours at 55° C. rise, 1,150 r.p.m., are used for the types 125 and 150 rock-dust distributors and are rated at 25 horsepower for 1 hour at 55° C. rise, 2,300 r.p.m., for the 250-volt type 350 rock-dust distributor. The 500-volt type 350 machine uses a Westinghouse type 93-SK motor rated 25 horsepower. The explosion-proof features of the types 103-SK and 93-SK motors are essentially the same as those of the type 53-SK motor described in connection with Approval No. 130. The end bells are held by 5/8 inch tap bolts, and the close fit through the inside bearing cap at one end to prevent the discharge of flames along the shaft is obtained by pressing a sleeve on the shaft of the type 103-SK motor to increase the diameter. The second shaft extension on motors with the double shaft extensions passes with a close fit through the commutator end outer bearing cap, thus making a similar con-

struction at both ends. Except for these differences and their size, the types 103-SK and 93-SK motors correspond very closely with the type 53-Sk motor.

Except for the 500-volt No. 350 machine, the control is a Westinghouse, push-button-operated, automatic, magnetic starter of the counter E.M.F. acceleration type with one step of starting resistance. It is wired for low-voltage protection. The parts consist of two single-pole line contactors, a single-pole accelerating contactor, a single-pole magnetic overload relay, the starting resistance, two fuses for the headlight circuit, and the push-button station. The two push buttons are operated by plungers from outside the control enclosure and serve to start and stop the motor.

The control enclosure consists of three main parts - a cast-steel base, a cylindrical case of 14-1/2 inches inside by 15 inches outside diameter steel pipe, and a cast-steel top or cover. The base and cover have slip fits over the cylindrical case, which is accurately machined for this purpose, and the ends of the case bear against shoulders machined inside the base and cover. Thus, a right-angle joint is formed at each end with wide metal surfaces between the parts. The assembly is held together by a single center bolt, which screws into a threaded hole through a boss at the center of the base casting and goes through a bored hole in a boss at the center of the top. A special cap nut with a skirt that fits the bore through the top casting screws on the end of the bolt to hold the parts in place. The nut is sealed to prevent tampering, and its long close fit makes a flamtight joint with the top casting. A key piece held by a small screw prevents the center bolt from turning in the bottom casting. The two push-button plungers have long, close fits through the top casting. The cables enter and leave the control compartment through asbestos-packed stuffing boxes in the base casting. All three stuffing-box nuts are designed for hose conduit that protects the cables.

The control for the 500-volt No. 350 machine is a push-button-operated magnetic starter of the time-limit acceleration type with two steps of starting resistance. It is connected for low-voltage protection. A push-button-operated switch is provided in the control for the headlight circuit, and two small fuses protect both the control and headlight circuits. The principal accessory parts consist of two single-pole line contactors, two single-pole accelerating relays (acting both as relays and contactors), a single-pole magnetic thermal overload relay, the starting resistance, the push-button station, the headlight switch, and two small fuses for the control and headlight circuit. The control enclosure is a rectangular box of welded 5/16-inch thick steel plate with one open side. This side is closed by a 1/2-inch thick flat steel plate held by eighteen 1/2-inch tap bolts that go into tapped bottomed holes in four 1-by-1-1/4 inch steel bars welded around the inside of the opening into the box. The bolts are secured by lock washers, and the wide metal surfaces in contact prevent discharge of flame through the joint. The plungers for operating the control pushbuttons and for the headlight switch are journaled with long close fits through bosses welded to the box. The stuffing boxes for the cables screw into holes tapped through a steel bar welded to the box. The stuffing-box gland nuts for the cables to

the motor and to the headlight are arranged to hold the hose conduits that protect the cables. The stuffing-box gland nut for the cable to the power supply has a bell mouth and an insulated strain clamp incorporated in its design. However, this cable is protected by hose conduit at the machine and is held by a clamp to the frame of the machine. The trailing cable is No. 3, two-conductor, round, rubber-clad cable; the cable to the motor is No. 3, three-conductor, round, rubber-clad cable and the headlight cable is No. 16, two-conductor, round, rubber-clad cord.

The Westinghouse type S headlight (b, fig. 2) consists of a cylindrical, boxlike, cast-iron body, into which is screwed a cover holding a glass 1/2-inch thick. The glass is held between lead gaskets by means of a ring fastened to the back of the cover by six screws. A seal prevents unauthorized persons from removing the cover. The lead entrance for the two-conductor cable to the headlight is sealed by plaster of paris in pipe fittings screwed into a tapped hole in the headlight body. The fittings are arranged to secure the end of the hose conduit that protects the cable. An extra lead entrance hole in the headlight body is closed by a screw plug welded in place.

The Mancha Storage Battery Locomotive Co. headlight used on the 250-volt machines was the same as that described for Approval No. 130. The Mancha headlight, type M, for the 500-volt No. 350 machine is a somewhat modified design. The cylindrical shell is cast iron with one end cast closed; the front cover or glass holder screws on the open end. The stuffing-box lead entrance for the cable is machined in a boss on the outside of the closed end. The screw cover is locked by a rod that passes through lugs cast on the case and is secured by a padlock. The end of the rod rests in a notch in the periphery of the screw cover. A flat glass 1/2-inch thick fits against a lead gasket in a machined seat in the screw cover. It is held by a ring that screws against it from the rear; a set screw prevents the ring from loosening. The wide surfaces in contact between the glass, the gasket, and the retaining ring result in a flametight joint. The internal parts are held by screws in tapped bottomed holes.

The headlight resistance compartment is a rectangular box of welded construction built of 1/4-inch plate steel with 1/2-by 1-inch steel bars welded around the top to provide for a wide metal-to-metal joint with the cover plate. The cover is 1/4-inch steel plate held by twelve 1/2-inch cap screws that go into holes tapped through the 1/2-by 1-inch flange bars outside the enclosure and secured by lock washers. The wide metal-to-metal cover joint is flametight. The cable from the controller and from the resistance box to the headlight is two-conductor, No. 16, round, rubber-clad cord. The cable entrances at the resistance are asbestos-packed stuffing boxes machined in fittings screwed into holes tapped through steel blocks welded to the bottom of the compartment. An unused hole is closed permanently by a screw plug welded in place. The stuffing-box nuts are arranged to secure the hose conduit that protects the cables.

American Mine Door Co.

Type H Rock-Dust Distributor

Approvals Nos. 180 and 180A

Approval No. 180 for the American Mine Door Co. 250-volt, type H, rock-dust distributor was issued October 30, 1929; Approval No. 180A for the 500-volt machine was granted January 17, 1930.

Figure 3 shows the American Mine Door Co. type H rock-dust distributor without the sheet-metal covers over the hopper and the motor and control section.

Motor a is a Westinghouse 113SK, 20 horsepower, 1,150-r.p.m. motor, with double-shaft extensions, one for the blower and the other for the agitator and conveyor drive.

The frame is made up of a piece of steel rolled to form a cylinder, after which the ends are welded together to form one continuous magnetic circuit. Steel forgings riveted to the frame constitute feet for the motor. The two bearing brackets are of cast iron. The front bracket encloses the commutator of the motor. It has four openings equally spaced, through which adjustment of the brushes can be made and the commutator inspected. These openings are threaded to take brass covers. During normal operation of the mechanisms these covers are secured with a seal to prevent unauthorized opening.

Each bracket is made with a lip that fits closely in the frame. Both also have a shoulder that fits against the end of the frame. Thus, a section through the joint between the bracket and frame shows a 90° change in direction in the path from the interior to the exterior of the motor. Eight cap screws secured with lock washers hold each bracket to the frame.

In addition to the four inspection openings in the front bracket, there is also a tapped hole for a stuffing box, through which a three-conductor cable enters the motor. Each bracket has an opening, into which a ball bearing fits. Two parts are necessary to complete each bearing housing. The inner housing cover or cap at each end has four studs screwed in it and secured by welding. These extend through holes in the bracket and are long enough to pass through the outer bearing cap, which is then held by nuts secured with lock washers. Each cap makes a step joint with the bracket; that is, it is machined so that one portion fits inside the opening for the bearing and another portion at right angles to it seats against the bracket. The inner caps fit the shaft closely enough so that, in conjunction with the bearings themselves, the passage of flame from explosion within the motor is stopped effectively. The eyebolt in the top of the frame is secured permanently by welding to guard against a through hole at this point.

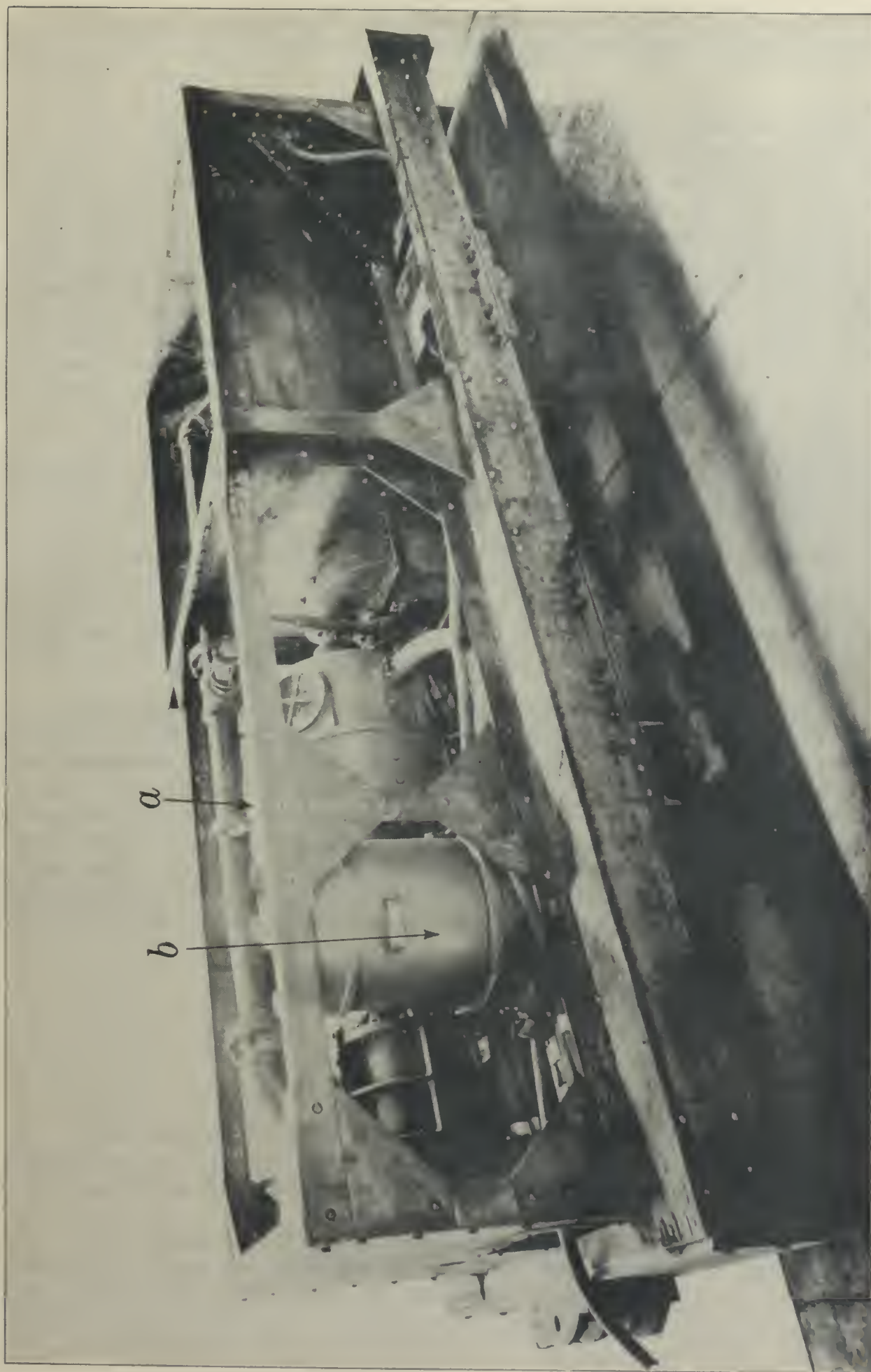


Figure 3.- American Mine Door Co. type H rock-dust distributor.

The explosion-proof features of the controller b are essentially the same as those described in connection with the first control under Approval No. 146.

The 250-volt control is a one-step magnetic starter of the counter E.M.F. type connected for low-voltage protection. It contains two single-pole line contactors, one accelerating contactor, one magnetic overload relay, the push-button station, the starting resistance, and two fuses for the headlight circuit. The 500-volt control is a two-step magnetic starter of the time-limit acceleration type connected for low-voltage protection. It contains two single-pole line contactors, two accelerating contactors, two accelerating relays, one magnetic overload relay, the push-button station, the starting resistance, and two fuses for the headlight circuit. The incoming positive line must be connected to the positive terminal of the control because the overload relay is single pole.

The 250-volt machine carries one Westinghouse type S headlight; the 500-volt machine has two headlights connected in series. This headlight is the same as that described for the Mine Safety Appliances Co. rock-dust distributor of Approval No. 146A, except that the cable enters through an asbestos-packed stuffing box, which screws into the body of the headlight. On 500-volt machines the cable to the second headlight comes from the first light, which it leaves through a second stuffing box. The extra tapped lead entrance hole in the second headlight and in the one headlight on 250-volt machines is closed by a screw plug permanently welded in place.

The cables from the control to the motor, from the control to the headlight (and between the two headlights on 500-volt machines) are protected by hose conduits well cleated in place. Round, rubber-clad, three-conductor No. 3 and two-conductor No. 16 cable and cord are used. The trailing cable to the power supply is two-conductor, No. 2, round, rubber-clad cable 25 to 300 feet long. The outer end of the stuffing-box nut is in the form of a bell mouth and also carries an insulated strain clamp to hold the cable. The trailing cable may be equipped with a fused trolley tap and rail clamp for connection to the power circuit.

Diamond Machine Co.

"Legrabon" 95-Volt Rock-Dust Distributor

Approval No. 183

The Diamond Machine Co. Legrabon rock-dust distributor, arranged for operation by a 95-volt storage-battery power truck, was approved February 4, 1930. Approval No. 183 was assigned to this design.

The Diamond Machine Co. 95-volt, Legrabon, rock-dust distributor, covered by Approval No. 183, is the same as that covered by Approval No. 131, except that a different motor and control are used.

The motor is a Westinghouse 113 SK, 12 horsepower, 1,400 r.p.m., 95-volt one with a single shaft extension. Its explosion-proof features are essentially the same as those of the motor for the American Mine Door Co. type H rock-dust distributor, Approvals No. 180 and 180A, except that only one of the outer bearing caps is pierced for a shaft extension. The control is a one-step magnetic starter of the counter E.M.F. type connected for low-voltage protection. It contains two single-pole line contactors, one accelerating contactor, a single-pole magnetic overload relay, the push-button station, and the starting resistance. The explosion-proof features of the control enclosure are essentially the same as those of the cylindrical control inclosure for the Mine Safety Appliances Co. rock-dust distributors covered by Approvals Nos. 146 and 146A. (A plug is welded in one of the three stuffing boxes to permanently close this opening as it is not used for the present machine.)

The cable from the control to the motor is a round, three-conductor, No. 3, rubber-clad cable protected by a hose conduit. The stuffing-box nuts are designed to secure the ends of the conduit. The cable to the power supply is a 40-foot length of 2-conductor, No. 2, round, rubber-clad cable protected by hose conduit throughout its length. At the control end the conduit is clamped to the frame of the machine. The cable enters the control through a stuffing box bell-mouth gland with an insulated clamp for the cable. At the outer end the cable has a plug for connection to a storage-battery power truck. The hose conduit is clamped to this plug and the cable is held by an insulated clamp to prevent stress on the connections.

Diamond Machine Co.

High-Pressure Rock-Dust Distributor

Approval No. 195

Approval No. 195 for the Diamond Machine Co. high-pressure rock-dust distributor was issued July 24, 1930. This machine has three explosion-proof inclosures, the motor, the control, and a field rheostat for varying the speed of the motor.

The motor is a Westinghouse 113 SK, 15 horsepower, 1,150 to 1,600-r.p.m., 230-volt motor with a single shaft extension. Its explosion-proof features are the same as those described for the motor for the American Mine Door Co. type H rock-dust distributor, Approvals Nos. 180 and 180A, except that only one of the outer bearing caps is pierced for a shaft extension.

The control is a one-step magnetic starter of the counter E.M.F. type connected for low-voltage protection. It contains two single-pole line contactors, one accelerating contactor, one single-pole magnetic overload relay, the push-button station, and the starting resistance. The explosion-proof features of the control enclosure are essentially the same as those for the cylindrical control enclosure described for the Mine Safety Appliances Co. rock-dust distributors of Approvals Nos. 146 and 146A. (The third stuffing box is arranged for a two-conductor, No. 16 cable to the field rheostat.)

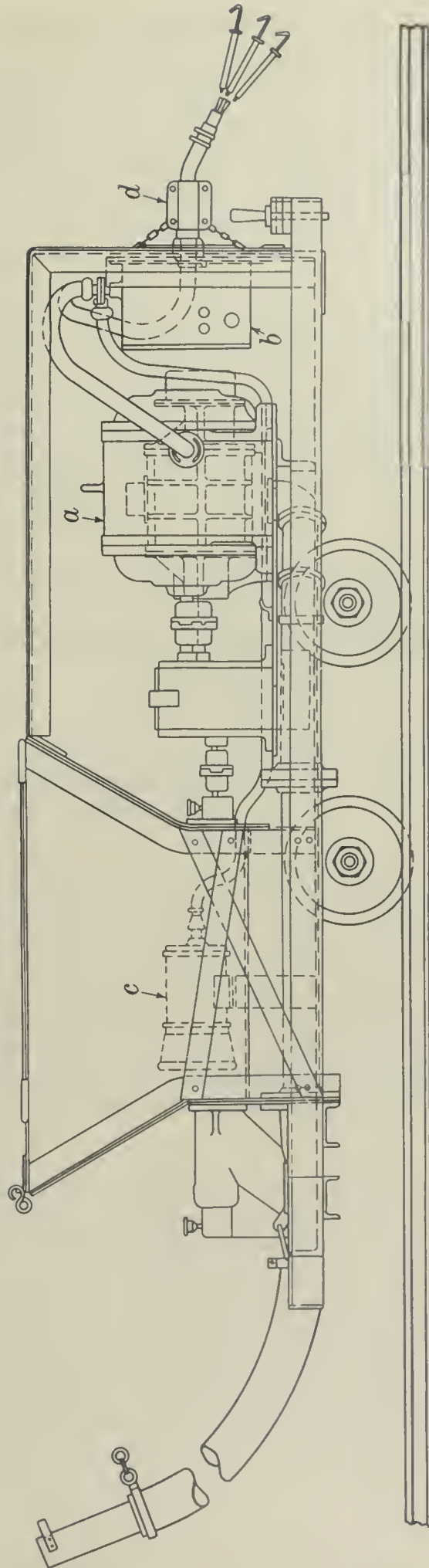


Figure 4.— Mine Safety Appliance Co. type 65 A. C. rock-dust distributor.

The field rheostat is manually operated to change the speed of the motor. It is enclosed in a rectangular, cast-iron case, with a 3/8-inch cast-iron cover held by sixteen 1/2-inch cap screws in holes bottomed in the case and secured by lock washers. The wide, flat, metal contact between the case and cover makes a flamtight joint. The operating shaft for the rheostat has a long close fit in a bronze bushing in a hole through the center of the cover. The two-conductor cable enters through an asbestos-packed stuffing box screwed into a boss and secured by a set screw. The stuffing-box nut is arranged to hold the hose conduit protecting the cable.

The connection from the control to the motor is by a round, rubber-clad, three-conductor, No. 3 cable protected by hose conduits. The stuffing-box nuts hold the ends of the conduit. The trailing cable is a two-conductor, No. 2, rubber-covered cable 25 to 200 feet long. The stuffing box at the control has a hell mouth and an insulated cable clamp for this cable; in addition, an insulated clamp secures the cable to the frame of the machine. The outer end of the cable may be equipped with a trolley tap fused to 125 amperes for connection to the power circuit.

Mine Safety Appliances Co.

Type 65 A. C. Rock-Dust Distributor

Approval No. 206

Approval No. 206 was granted the Mine Safety Appliances Co. on November 12, 1930, for their type 65, 220-volt, rock-dust distributor, which is an alternating-current machine.

Figure 4 is an elevation showing the positions of the electrical accessories and wiring. At a is a Westinghouse Electric & Manufacturing Co. type OS, frame 370-C, 220-volt, 3-phase, 5 horsepower, 1,150-r.p.m., squirrel-cage induction motor; at b is a Westinghouse type TX-33, 5 horsepower, 230-volt, 3-phase, alternating-current line starter; at c is a Mancha Storage Battery Locomotive Co. type F headlight. The starter consists of a three-pole magnetic line contactor, a two-pole thermal overload relay, three 30-ampere main-line fuses to furnish short-circuit protection, a push-button station, two 6-ampere fuses, and a two-pole snap switch for the headlight circuit.

Connection from the starter to the motor is by a three-conductor, No. 10, rubber-clad cable, and connection from the control to the headlight is by a two-conductor, No. 16, rubber-clad cord. The trailing cable is a three-conductor, No. 10, rubber-clad cable 50 to 300 feet long, which may be connected to the power circuit by three fused trolley taps if the connection is made in pure intake air. The cable entrances to the motor, control, and headlight are asbestos-packed stuffing boxes with gland nuts arranged to secure the ends of hose conduits that protect all the wiring on the machine. Insulated strain clamp, d, attached to the machine by short chains, holds the trailing cable to prevent strain on the cable entrance and connections at the starter.

The Mancha Storage Battery Locomotive Co. headlight is the same as that described for Approval No. 130.

The starter compartment is a steel box constructed of 1/4-inch steel plates welded together, with an open side. Four 1-1/4 by 1-inch steel bars are welded around the inside of the opening to provide for a wide metal-to-metal joint with the 3/8-inch steel plate cover. The cover is held by fourteen 1/2-inch cap screws in tapped bottomed holes in the bars. Lock washers prevent the screws from loosening. The stuffing boxes for the main cables are machined in steel blocks welded to the compartment; that for the headlight cable is in a fitting that screws into a steel block welded to the compartment. The long bearing fits required to prevent discharge of flame along the push-button plungers and headlight-switch shaft are in steel bearing blocks welded to the compartment.

The motor frame is cast iron and cylindrical, with the feet cast integral. It is machined at the ends for the joints with the cast-iron bearing brackets that close the ends of the cylinder and carry the ball bearings for the shaft. The joints are of the rabbet type. A machined lip on the rim of the bracket fits into the bore of the frame, while a shoulder on the rim is held against the end of the frame by eight 3/8-inch tap bolts; a right-angle joint is thus formed with wide metal surfaces in contact between the parts. The bolts go into tapped bottomed holes in the frame and are secured by lock washers. The ball bearings fit bored holes through the centers of the bearing brackets and are each held in place by two bearing caps, one inside and one outside. Four 3/8-inch studs welded to the inside cap pass through holes in the bearing bracket and in the outer bearing cap to hold the parts. The nuts are secured by lock washers. The bearing caps make right-angle joints with the bearing brackets, with wide metal surfaces in contact between parts. To prevent discharge of flames along the shaft, it has a close running fit through the inside bearing caps for a considerable distance. The outside caps furnish additional protection. The grease holes should be closed to obtain full benefit from the outside caps. The eye bolt in the top of the motor frame is permanently welded in place. The wires enter the motor in the form of a three-conductor, No. 10, rubber-clad cable through an asbestos-packed stuffing box. The stuffing-box screws into a tapped hole through the side of the frame and is secured by a small screw. The stuffing-box nut is arranged to secure the hose conduit that protects the cable.

Mine Safety Appliances Co.

Type 85 Rock-Dust Distributor

Approval No. 279

The Mine Safety Appliances Co. type 85 rock-dust distributor was approved for 80-volt service on February 14, 1935. No changes have been made by extension of the approval.

The various accessory parts of the Mine Safety Appliances Co. type 85 rock-dust distributor covered by Approval No. 279 are carried on a four-wheeled truck. The blower and the feed mechanism in the hopper are driven by a 3 horsepower, 80-volt motor through a gear box. The controller and the headlight are placed on opposite sides of the hopper near the opposite end of the truck from the motor and blower.

Power is supplied from a storage-battery power truck through a No. 8, concentric, rubber-clad cable protected by hose conduit and held by a strain clamp attached to the machine by a chain. An insulated clamp on the cable inside the control compartment also holds the cable. Connection from the control to the motor is by a No. 8, concentric, rubber-clad cable protected by hose conduit. The headlight cable is two-conductor, No. 16, rubber-clad cord protected by hose conduit.

The headlight is the Mancha Storage Battery Co. type M design described in connection with the Mine Safety Appliances Co. 500-volt, type No. 350, rock-dust distributor, or the Mancha headlight described in connection with Approval No. 130.

The motor is a Westinghouse Electric & Manufacturing Co. type 33-SK, 3 horsepower, 80-volt design. The explosion-proof features of the type 33-SK motor are essentially the same as those of the type 53-SK motor described in connection with Approval No. 130. The end bells are held by 3/8-inch tap bolts and the studs that hold the bearing caps are 3/8-inch diameter at both ends. The cable stuffing box is made suitable for the No. 8 concentric cable on this rock-dusting machine. Otherwise, except for its smaller size, the type 33-SK motor is similar to the type 53-SK.

The control is a Westinghouse Electric & Manufacturing Co. Class 8508 across-the-line magnetic, push-button-operated starter consisting of a two-pole magnetic contactor, a thermal overload relay, a push-button station, and a headlight switch. A fuse is included, connected in the positive side of the circuit, to furnish short-circuit protection, and two small fuses are connected to protect the headlight and control wiring. The control compartment is a rectangular box of welded 1/4-inch thick sheet steel with 1/2 by 1-1/2-inch steel flange bars welded around the outside of the open front to provide for the wide metal-to-metal joint with the cover. The cover is 11/32-inch steel plate bolted to the compartment by fourteen 1/2-inch bolts going through the flange bars outside of the compartment. The nuts are secured by lock washers. The wide flat metal surfaces of the cover and box held in contact by the bolts make the joint flametight.

Two plungers for operating the control and two for the headlight switch are journaled with long close fits in holes through four steel bosses welded to the cover. The No. 8, concentric, rubber-clad cable from the power supply and to the motor enter the control through asbestos-packed stuffing boxes machined in steel bosses welded to the case. An insulated clamp is placed on the trailing cable just inside the stuffing box to prevent the cable being pulled out or strain put on the connections. The headlight cable,

a No. 16, two-conductor, rubber-clad cord enters the control through an asbestos-packed stuffing box in a fitting that screws into a hole tapped through a steel pad welded to the control case. All the stuffing-box gland nuts are arranged to secure the hose conduits used to protect the cables.

CONCLUSION

The thorough rock-dusting of a bituminous-coal mine stands as the most important method for the prevention of extensive mine explosions propagated by coal dust. Consequently, it is highly essential that a piece of equipment designed for use in applying rock dust to prevent such dust explosions shall not at the same time add to the possibilities of igniting gas that might, in turn, cause a dust explosion. This hazard is reduced to a minimum in the rock-dust distributing machines described in this paper.

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UNITED STATES BUREAU OF MINES
JOHN W. FINCH, DIRECTOR

REPORT OF INVESTIGATIONS

ANALYSES OF CRUDE OILS FROM SOME FIELDS OF MICHIGAN



BY

E. L. GARTON

REPORT OF INVESTIGATIONS

DEPARTMENT OF THE INTERIOR - BUREAU OF MINES

ANALYSES OF CRUDE OILS FROM SOME FIELDS OF MICHIGAN^{1/}

By E. L. Garton^{2/}

INTRODUCTION

Michigan is becoming one of the important oil-producing States of the country. The amount of crude oil produced within its boundaries has increased from only 4,000 barrels in 1925^{3/} to nearly 16 million barrels in 1935.^{4/} The Bureau of Mines method of crude-oil analysis and classification indicates that the oils from the several producing fields are predominantly paraffin-intermediate-base, wax-bearing crudes. In general, they will yield 25 to 35 percent of highly paraffinic gasoline and about the same amount of kerosene and gas-oil combined, 12 to 15 percent of lubricating oil stock, and 20 percent of residuum with a carbon residue of about 6 to 7 percent. In addition, there appears to be some correlation between certain properties of the crude oils and both the geographical location and geological structure of the oil-producing areas within the State. These general conclusions are based on the analytical data presented in this report, which is one of a series of papers that record the general characteristics of crude oil from the many producing fields of the United States.

This report discusses briefly the oil fields of Michigan and the characteristics of the crude oils produced from them. The analyses of 15 samples are given. These samples were obtained from 13 fields in 8 counties in the lower peninsula and represent crude oils produced from the Berea sandstone, Upper Traverse and Traverse limestones, and the

^{1/} The Bureau of Mines will welcome reprinting of this paper provided the following footnote acknowledgment is used: "Reprinted from U. S. Bureau of Mines Report of Investigations 3346."

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^{3/} Hopkins, G. R., and Coons, A. B., Crude Petroleum and Petroleum Products: Minerals Yearbook, Statistical Appendix, 1935, p. 365.

^{4/} White, A. G., Hopkins, G. R., and Breakey, H. A., Crude Petroleum and Petroleum Products: Minerals Yearbook 1936, Bureau of Mines, p. 697.

Dundee and Trenton limestone formations. They were collected from May 10, 1936, to June 8, 1936, and therefore represent current production; the analyses, therefore, are on a comparable basis. All analyses were completed by August 15, 1936. The oldest well covered by these samples (no. 2) was completed during November 1927, and six of the samples were from wells completed in 1936. Analysis of a sample from the Bradford Third Sand of the Bradford field, McKean County, Pa. (no. 16), also is included for comparison.

ACKNOWLEDGMENTS

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Dr. R. A. Smith, State geologist, and F. R. Frye, petroleum engineer, both of the Geological Survey Division, Department of Conservation, of the State of Michigan, helped in obtaining representative samples, and their cooperation in making this report possible is given special acknowledgment.

MICHIGAN OIL FIELDS

Michigan has an area of 37 million acres, of which approximately 20 million acres have been classified as possible oil-producing areas. Only 5 million acres have been prospected, but it is possible that a large productive area may yet be found.^{5/} Oil was discovered accidentally in Michigan in 1865^{6/} in wells drilling for salt water, but drilling was more or less sporadic until 1926.

The Saginaw field on the east side of the southern peninsula, was discovered in 1924.^{7/} Four years later oil was produced in commercial quantities at Muskegon, adjacent to Lake Michigan, and within the next 2 years Mount Pleasant (Greendale Township), Leaton, Vernon, and Porter had been added to the list of producing fields, to be followed in 1933 with the West Branch and Edmore fields. During 1935, several more fields were found, among which were the Beaverton, Birch Run, Crystal, and Denver Township. The Monroe County field was added to the producing areas in 1936. The crude oil produced in Michigan during 1935 was more than that produced in any other single State east of the Mississippi River.^{8/}

^{5/} Ziegenhain, W. T., Operators in Michigan Face Troubles When Drilling to Deeper Strata: Oil and Gas Jour., vol. 35, no. 16, Sept. 3, 1936, p. 26.

^{6/} Egloff, Gustav, and Nelson, Edwin F., Survey of Michigan Crude Analyses Shows Necessity for Cracking: Oil and Gas Jour., vol. 33, no. 45, March 28, 1935, p. 62.

^{7/} Ziegenhain, W. T., work cited in footnote 5.

^{8/} Elliott, Paul A., Michigan Field Report: Annual Review, Oil and Gas Jour., vol. 34, no. 37, Jan. 30, 1936, p. 124.

showing a 50-percent increase over the 1934 figure for number of wells drilled and quantity of crude oil produced. At the end of 1935, Michigan had a total of nearly 1,200 producing wells and had produced more than 1 million barrels of crude oil for each month of the year. This high rate of production continued through March 1936, marking a period of 15 months with more than 1 million barrels of crude oil produced each month.^{9/} Since that time total production of more than 1 million barrels per month has been obtained only during May and July 1936. The last available figures (Monthly Petroleum Statement 153, Jan. 13, 1937) show that production for October and November 1936 was 908,000 and 850,000 barrels, respectively.

The Crystal field, Montcalm County, has proved to be one of the largest oil fields in Michigan to date. The discovery well, brought in during March 1935^{10/} with a potential production of 5,000 barrels per day, was the largest "wildcat" well in the 10 years of oil-well drilling in Michigan. During the first 18 months of operation of this field, 5,500,000 barrels of crude oil was produced.^{11/} Production in this field is obtained from the Dundee limestone formation. The decline in the rate of production in this field has been considerably more rapid than was anticipated.

The production of the Crystal field is "spotty" and has a strong water drive and very little gas pressure.^{12/} The water drive is much greater in this field than in other Michigan fields and requires the maintenance of a considerable back pressure to prevent the invasion of water into the producing formation.^{13/} The peak production for this field, approximately 26,000 barrels per day, was obtained during October 1935.^{14/}

Newman^{15/} states that the West Branch field of Ogemaw County at present is of less importance than other Michigan fields in production of petroleum but that the field has geological interest and is important in the development of the industry outside the Central Basin. The field, approximately 9 miles long and nearly 1 mile wide, produces from both the Traverse and Dundee limestones. Development has been slow, and by the

^{9/} Hopkins, G. R., Crude Petroleum and Petroleum Products: Monthly Petroleum Statement Nos. 129-150, Bureau of Mines.

^{10/} Elliott, Paul A., Michigan Field Report: Oil and Gas. Jour., vol. 33, no. 46, Apr. 4, 1935, p. 165.

^{11/} Ziegenhain, W. T., work cited in footnote 5.

^{12/} Eddy, G. E., Geology of the Crystal Oil Field, Montcalm County, Michigan: Geological Survey Division, Department of Conservation, State of Michigan, Progress Report no. 1, July 1936.

^{13/} Weekly Staff Report, Michigan Field Report: The Oil Weekly, vol. 78, no. 10, Aug. 19, 1933, p. 69

^{14/} Eddy, G. E., work cited in footnote 12.

^{15/} Newman, E. A., Geology of Ogemaw County and West Branch Oil Field: State of Michigan, Geol. Survey Div., Dept. of Conservation, Progress Report 2, August 1936.

latter part of July 1936 only 138 oil wells had been completed. One gas well producing from the Traverse limestone has been completed. Samples 4 and 8 were obtained from this field on May 14, 1936. Sample 4 was from a well completed in December 1933 and sample 8 was from a well completed in April 1936.

The Beaverton field, Gladwin County, opened in 1935, was extended to the northwest by a well drilled in sec. 3, T. 17 N., R. 2 W., with production from the Dundee limestone.^{16/} This well was completed in March 1936, and sample 14 was obtained from it May 25, 1936.

An extension of an old southeastern Michigan field, first tested in 1888, was indicated by a well in sec. 19, T. 6 S., R. 6 E., in Dundee township of Monroe County with production from the Trenton limestone.^{17/} This well, completed during April 1936, was the largest in Michigan's oldest oil territory. Sample 15 was obtained from it May 10, 1936.

One of the largest wells completed in the Birch Run field, Saginaw County, was in sec. 20, T. 10 N., R. 6 E., and produced from the Berea sandstone.^{18/} This well was completed in January 1936, and sample 1 was obtained from it May 16, 1936.

The discovery well of Saginaw field, near the city of Saginaw, on Saginaw Bay, Lake Huron, was completed in the Berea sandstone at a depth near 1,900 feet with an initial production of about 20 barrels a day.^{19/} Sample 2 was obtained from a well in sec. 10, T. 12 N., R. 4 W., completed in November 1927 with an initial production of 10 barrels a day from the Berea sandstone. This sample was obtained June 5, 1936.

Fields within the Central Michigan area from which samples were obtained are Mount Pleasant (Greendale Township), Leaton, Vernon, Porter, and Denver Township. This area, which is approximately 25 miles wide and 40 miles long, is regarded as a single structural province.^{20/}

The discovery well of the Mount Pleasant field, completed during February 1928, had an initial production of 30 barrels, which was increased to 100 barrels by drilling deeper into the sand. One large well of this field was brought in during May 1931 with an initial production

^{16/} Elliott, Paul A., Michigan Field Report: Oil and Gas Jour., vol. 34, no. 47, Apr. 9, 1936, p. 56.

^{17/} Elliott, Paul A., Michigan Field Report: Oil and Gas Jour., vol. 34, no. 48, Apr. 16, 1936, page 183.

^{18/} Elliott, Paul A., Michigan Field Report: Oil and Gas Jour., vol. 34, no. 36, Jan. 23, 1936, p. 73.

^{19/} Ziegenhain, W. T., work cited in footnote 5.

^{20/} U. S. Geological Survey Staff, Geology and Occurrence of Petroleum in the United States: Petroleum Investigation Hearings Before a Subcommittee on Interstate and Foreign Commerce, House of Representatives, 73d Congress (Recess), on H. Res. 441, Sept. 17-22, 1934, part 2, pp. 940-946.

of over 800 barrels. Sample 11 was obtained May 22, 1936, from a well of the Greendale Township extension of this field. This well, in sec. 10, T. 14 N., R. 2 W., was completed in September 1931 and produces from the Dundee limestone. Samples 12 and 13 were obtained from wells in the Leaton and Vernon fields, respectively, which were completed in 1930, and sample 10 was obtained from a well in the Porter field, which was completed in 1934. All three wells produce from the Dundee limestone and were sampled in May 1936. Sample 6 was obtained May 27, 1936, from a well in the Denver Township field that produces from the Traverse limestone and was completed February 3, 1936.

Although drilling in the Muskegon field of Muskegon County may have been done as far back as 1870, the first commercial production was obtained at the close of 1927 and during 1928. The first commercial well was brought in with an initial production of 300 barrels per day from the Traverse limestone.^{21/} By the close of 1928, production had also been obtained from the Dundee limestone. This field reached peak daily production of 18,570 barrels during August 1929 but declined to 1,160 barrels by May 1932. The producing formations of this field are the Dundee, Upper Traverse, Lower Traverse, and Upper Monroe limestones. Two samples from this field are included in the analyses of this report. Sample 3 is from a well in sec. 9 and sample 7 is from a well in sec. 8, both wells in T. 10 N., R. 16 W. The wells were completed in 1928 and produced from the Upper Traverse and Dundee limestones, respectively.

DISCUSSION OF ANALYSES

Base of a Crude Oil

In order to assist in the detailed discussion of these crude-oil analyses, a brief description of the method used by the Bureau of Mines in classifying crude oils is given. It has become general practice to classify crude oils according to "base". From the three groups used originally, the classification has been extended by the addition of four intervening groups, making a total of seven.^{22/}

In brief, this method of evaluating "base" depends upon the properties of two key fractions - the fraction distilling between 250° C. and 275° C. (482°-527° F.) at atmospheric pressure and known as key fraction 1, and the fraction distilling between 275° C. and 300° C. (527°-572° F.) at 40 mm absolute pressure and known as key fraction 2.

The gravity of key fraction 1 represents the lighter fractions of the oil, whereas the gravity of key fraction 2 represents the heavier

^{21/} U. S. Geological Survey Staff, work cited in footnote 20.

^{22/} For a more detailed discussion of this method see Lane, E. C., and Garton, E. L., Base of a Crude Oil: Report of Investigations 3279, Bureau of Mines, 1935, 12 pp.

distillates. The following tabulation summarizes the range of gravities and corresponding classification:

Key Fraction 1

When A.P.I. gravities are:	40.00 or lighter	33.10-39.90	33.00 or heavier
Classification is:	Paraffin	Intermediate	Naphthene

Key Fraction 2

When A.P.I. gravities are:	30.00 or lighter	20.10-29.90	20.00 or heavier
Classification is:	Paraffin	Intermediate	Naphthene

Table 1 lists the corresponding key fractions for each of the samples included in this report and gives the "base" of each sample, and table 2 summarizes some of the properties of these samples. Figure 1 shows a number of the characteristic properties graphically. The light dotted lines connecting these points do not form a functional curve but serve only to connect all points representing the same property for each of the 15 crude oils from Michigan fields and the Bradford sample included for comparison. Reference to figure 1 will help materially in understanding the following discussion.

Considered as a whole, the properties of these crude oils are not widely different. The two classed as intermediate (samples 4 and 5) are very near the dividing line between paraffin and intermediate (for the light fractions), while the single paraffin base oil (sample 15) is not greatly above the break between intermediate and paraffin (for the heavy fractions). Nevertheless, a casual inspection of certain of the properties would give the impression that there was considerable haphazard variation in certain characteristics. A close study of the data, however, indicates certain rather definite trends. Samples 7 to 14, inclusive, all produced from the Dundee limestone, show similarity in all of the characteristic properties, with a few exceptions that appear to have some connection with the geographical location of the producing area. Thus, sample 7, which shows an abnormally high sulphur content, is from the Muskegon field, and sample 3, from the Upper Traverse limestone in the same field, also has a sulphur content above the average. Sample 8, which is the only other one with a sulphur content much above the average, is from West Branch field of Ogemaw County, and somewhat northeast of the other Dundee limestone samples (with exception of no. 7) that originate in or are adjacent to the Central Michigan area.

These same samples (7 and 8) also have certain other exceptions that differentiate them somewhat from the other Dundee limestone samples. They both appear to be deficient in very light fractions, a deficiency that is shared by the companion samples 3 and 4 produced from the Traverse limestone in the Muskegon and West Branch fields, respectively. This deficiency of light fractions is reflected in crude-oil gravity and gravity of the gasoline fraction but does not appreciably change the

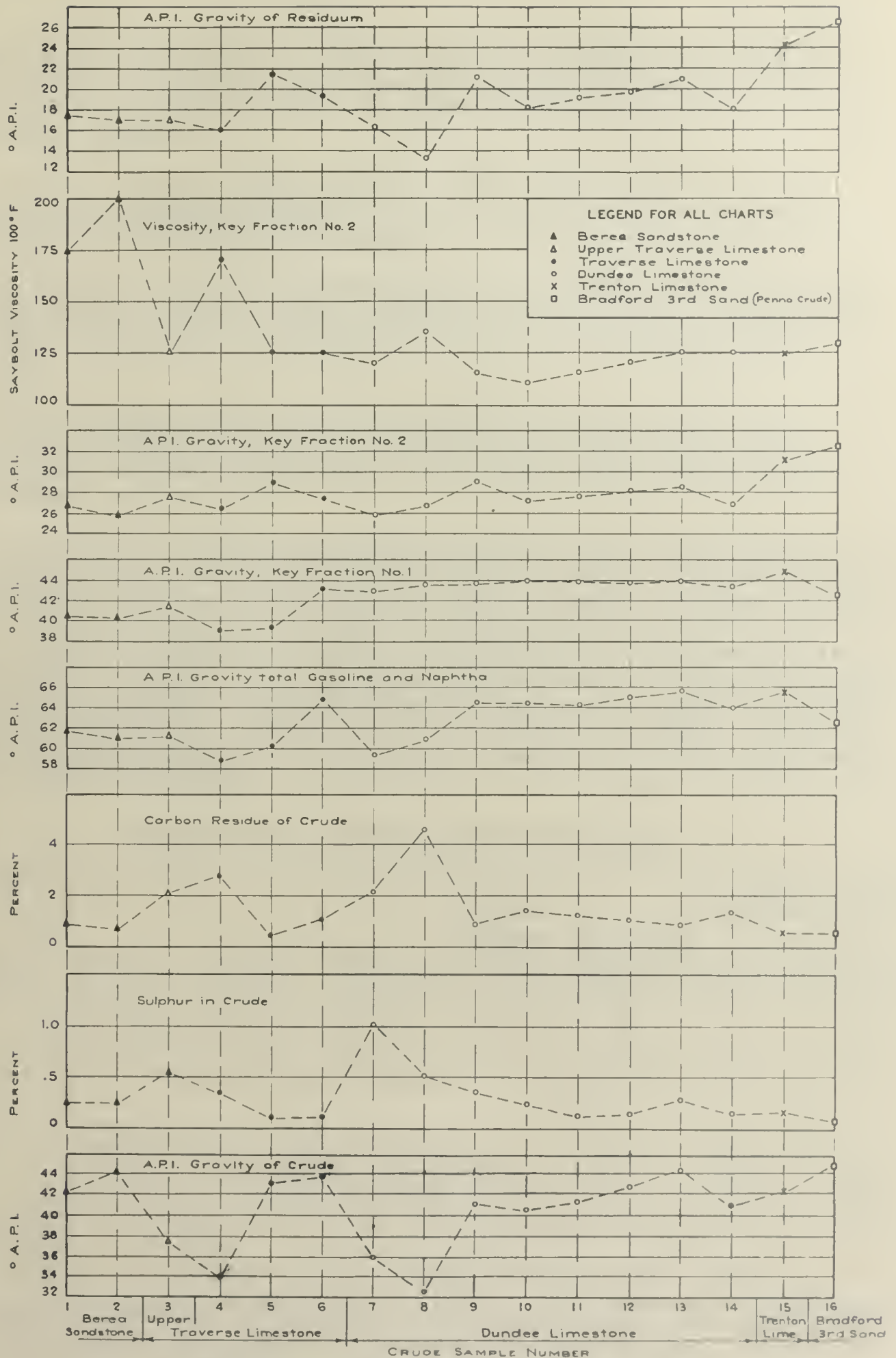


FIGURE 1-RELATIONSHIPS BETWEEN THE CHARACTERISTICS OF MICHIGAN CRUDE OILS AND THEIR PRODUCING FORMATIONS.

character of key fraction 1. Further, these two samples, although they have higher percentages of residuum, have lower A.P.I. gravities for the residuum and higher percent of carbon residue than any of the Dundee production sampled.

Both of the West Branch samples, 4 and 8 (especially no. 4, from the Traverse limestone), exhibit viscosities that are above the normal for any of the other limestone samples.

Sample 15, from the Trenton limestone is definitely more paraffinic throughout than any of the other Michigan samples analyzed. This paraffinicity is reflected in the gravities of the key fractions and the residuum.

The two samples from the Berea sandstone (numbers 1 and 2) are interesting chiefly in that key fractions 2 have definitely higher viscosities, with A.P.I. gravities only slightly lower than those for the Dundee limestone oils.

Newcombe,^{23/} in discussing analyses made on samples obtained from Central Michigan areas during 1930-31, states in part: "The Dundee and Upper Monroe crudes from the different pools of Central Michigan seem to have similar properties, but there is a slight variation from pool to pool toward the center of the basin." The properties he cites as indicating this trend are A.P.I. gravity of the crude oil, the percent sulphur, and gasoline recovery. The present set of samples collected during 1936 not only confirms the existence of this trend but also emphasizes variations of other properties not discussed by Newcombe. The following table shows these variations:

Properties of Central Michigan crude oils

Location (field)	Gravity, °A.P.I.		Sulphur, percent		Gasoline, percent		Carbon residue of crude, percent	Residuum gravity, °A.P.I.	Key fraction 1 gravity, °A.P.I.	Key fraction 2 gravity, °A.P.I.
	N1/ B1	B2/ B	N	B	N	B	B	B	B	B
Porter	40.4	40.6	0.18	0.26	33.0	30.8	7.2	18.2	43.8	27.1
Greendale Township, Sec. 10 ...	42.0	41.5	.11	.13	33.5	30.9	6.2	19.2	43.8	27.5
Leaton	43.0	43.0	.15	.16	35.0	33.9	5.6	19.7	43.6	28.0
Vernon	44.5	44.1	.24	.30	36.0	34.9	5.0	21.0	43.8	28.4

1/ Figures given by Newcombe.

2/ Bureau of Mines figures taken from analyses included in this report.

23/ Newcombe, Robert B., Oil and Gas Fields of Michigan: State of Michigan, Geol. Survey Div., Dept. of Conservation, Pub. 38, Geol. Series 32, June 1933, pp. 180-187.

The notable exception to this progressive change in properties is the high sulphur content of the Porter sample. The amounts of gasoline shown by the two sets of analyses given in the above table do not agree, as they were not calculated on the same basis, but both show the same trend.

Figure 1 also enables a comparison to be made between Michigan crude oils and a typical Pennsylvania crude oil produced from the Bradford third sand. Sulphur and carbon residue values, except for certain samples already noted as being out of line, are in general only slightly higher than the figures for the Bradford crude oil. The high paraffinicity of the light fractions from many Michigan oils is indicated by the A.P.I. gravities of total gasoline and naphtha, and key fraction 1, which in most of the samples are higher than similar values for the Pennsylvania crude oil. However, in the heavier ends the condition is reversed, and the higher A.P.I. gravities of key fraction 2 and the residuum indicate the reason why present preference is for oils of Pennsylvania type for lubricating-oil manufacture.

From a commercial point of view it appears that 25 to 35 percent of highly paraffinic straight-run gasoline may be obtained. According to Conine,^{24/} the high paraffin content of this gasoline is shown by the fact that straight-run gasoline from Texas crude oils contains 66.0 to 70.4 percent pure paraffin hydrocarbons compared with 69.8 to 74.9 for Oklahoma crude oils, 73.6 to 92.3 percent for Pennsylvania oils, and over 85 percent for gasoline from Michigan crude oils. This straight-run gasoline has an actane rating varying from 15 to 30, so that cracking or reforming is necessary to bring the final product to current octane requirements. The straight-run gasoline, when not cracked or reformed, has widespread use as naphtha and cleaning solvent.^{25/}

The kerosene and gas-oil stock, amounting to 25 to 35 percent, is suitable for cracking stock and diesel fuel. Some of the viscous portions, comprising 12 to 15 percent, should be suitable for manufacture of lubricating oils. The residuum, generally about 20 percent, is waxy, has low carbon residue, and should make good cracking stock.

SUMMARY

Samples of crude oil from thirteen Michigan fields, including production from the Berea sandstone and Traverse, Dundee, and Trenton limestones, have been analyzed.

The samples from the Dundee limestone, are similar to each other with certain exceptions. These differences appear in production from this formation outside of what is usually termed the Central Michigan area. The properties of certain crude oils produced within the area

^{24/} Conine, R. C., Outlook for Michigan Refiners is Question of Cracking: Oil and Gas Jour., vol. 34, no. 7, July 4, 1935, pp. 12-13.

^{25/} Newcombe, Robert B., work cited in footnote 23.

are much alike, but show some variation depending upon the distance of the producing pool from the center of the basin.

The Traverse limestone samples show local variation but in general are not materially different from the Dundee production.

The samples from the Muskegon and West Branch fields show considerable variation from the other limestone production in a number of properties.

The Berea sandstone crude oil is characterized by relatively high viscosity of the vacuum fractions.

As a whole, Michigan crude oils are highly paraffinic in the lighter fractions, and the heavier fractions, which are all wax-bearing, should be good lubricant stock, especially for the manufacture of less viscous oils.

TABLE 1. - Summary of data indicating the base of the crude oil

Sample	Field	County	Formation	Depth, feet
1...	Birch Run	Saginaw	Berea sandstone	1,525-1,545
2...	Saginaw	do	do	1,843-1,858
3...	Muskegon	Muskegon	Upper Traverse limestone	1,661-1,667
4...	West Branch	Ogemaw	Traverse limestone	1,711-1,721
5...	Edmore	Montcalm	do	3,025-3,097
6...	Denver Township	Midland	do	2,975-3,104
7...	Muskegon	Muskegon	Dundee limestone	2,035-2,062
8...	West Branch	Ogemaw	do	2,499-2,677
9...	Crystal	Montcalm	do	3,216-3,221
10...	Porter	Midland	do	3,365-3,436
11...	Mount Pleasant (Green- dale Township)	do	do	3,506-3,567
12...	Leaton	Isabella	do	3,604-3,662
13...	Vernon	do	do	3,720-3,724
14...	Beaverton	Gladwin	do	3,837-3,908
15...	Monroe County	Monroe	Trenton limestone	2,072-2,108
16...	Bradford (Pennsylvania)	McKean	Bradford third sandstone	1,771-1,881

Continued --

TABLE 1. - Summary of data indicating the base of the crude oil (Continued)

Sample	Fraction distilling at atmospheric pressure between 250° and 275° C. ^{1/}		Fraction distilling at 40 mm pressure between 275° and 300° C. ^{2/}		Cloud point, °F.	Indication of wax	Base of crude
	Sp. grav.	°A.P.I.	Sp. grav.	°A.P.I.			
1...	0.823	40.4	0.895	26.6	80	Present	Paraffin-intermediate
2...	.824	40.2	.900	25.7	75	do	do
3...	.819	41.3	.890	27.5	75	do	do
4...	.830	39.0	.896	26.4	80	do	Intermediate
5...	.829	39.2	.883	28.8	90	do	do
6...	.811	43.0	.891	27.3	75	do	Paraffin-intermediate
7...	.812	42.8	.900	25.7	75	do	do
8...	.809	43.4	.895	26.6	80	do	do
9...	.809	43.4	.882	28.9	80	do	do
10...	.807	43.8	.892	27.1	75	do	do
11...	.807	43.8	.890	27.5	80	do	do
12...	.808	43.6	.887	28.0	80	do	do
13...	.807	43.8	.885	28.4	75	do	do
14...	.810	43.2	.894	26.8	80	do	do
15...	.803	44.7	.870	31.1	90	do	Paraffin
16...	.814	42.3	.864	32.3	90	do	do

^{1/} Key fraction 1.^{2/} Key fraction 2.

TABLE 2. - Summary of properties of crude oils from 13 fields of Michigan and 1 Pennsylvania field

Sample	Field	County	Formation	Depth, feet
1...	Birch Run	Saginaw	Berea sandstone	1,525-1,545
2...	Saginaw	do	do	1,843-1,858
3...	Muskegon	Muskegon	Upper Traverse limestone	1,661-1,667
4...	West Branch	Ogemaw	Traverse limestone	1,711-1,721
5...	Edmore	Montcalm	do	3,025-3,097
6...	Denver Township	Midland	do	2,975-3,104
7...	Muskegon	Muskegon	Dundee limestone	2,035-2,062
8...	West Branch	Ogemaw	do	2,499-2,577
9...	Crystal	Montcalm	do	3,216-3,221
10...	Porter	Midland	do	3,365-3,436
11...	Mount Pleasant (Green-dale Twp.)	do	do	3,506-3,567
12...	Leaton	Isabella	do	3,604-3,662
13...	Vernon	do	do	3,720-3,724
14...	Beaverton	Gladwin	do	3,837-3,908
15...	Monroe County	Monroe	Trenton limestone	2,072-2,108
16...	Bradford (Pennsylvania)	McKean	Bradford 3d sandstone	1,771-1,881

TABLE 2. - Summary of properties of crude oils from 13 fields of Michigan and 1 Pennsylvania field (Continued)

Sample	Field	Properties of crude oil ^{1/}					Gasoline or naphtha fraction	
		Gravity, °A.P.I.	Percent sulphur	Percent carbon residue	Viscosity at 100° F., seconds	Pour point	Percent	Gravity, °A.P.I.
1...	Birch Run	42.3	0.25	0.9	36	Below 5° F.	38.6	61.8
2...	Saginaw	44.3	.24	.7	34	do	44.0	61.0
3...	Muskegon	37.4	.56	2.1	44	do	27.9	61.3
4...	West Branch	34.0	.35	2.8	52	do	28.0	58.9
5...	Edmore	43.2	.11	.5	35	do	41.1	60.2
6...	Denver Township	43.8	.12	1.1	36	do	36.1	64.8
7...	Muskegon	36.0	1.03	2.2	45	do	26.0	59.5
8...	West Branch	32.7	.52	4.6	60	do	22.0	60.8
9...	Crystal	41.3	.37	.8	37	do	36.5	64.5
10...	Porter	40.6	.26	1.5	38	do	30.8	64.5
11...	Mount Pleasant (Greendale Twp.)	41.5	.13	1.3	39	do	30.9	64.2
12...	Leaton	43.0	.16	1.1	37	do	33.9	65.0
13...	Vernon	44.1	.30	.9	39	do	34.9	65.6
14...	Beaverton	41.3	.16	1.4	38	do	32.7	63.9
15...	Monroe County	42.1	.18	.6	43	do	28.9	65.3
16...	Bradford (Pennsylvania)	45.2	Less than 0.1	.3	39	do	37.6	62.6

^{1/} Pour point of all samples, below 5° F.

Continued

TABLE 2. - Summary of properties of crude oils from 13 fields of Michigan and 1 Pennsylvania field (Continued)

Sample	Field	Viscosity at 100° F. of vacuum fractions			Residuum		
		225-250°C., seconds	250-275°C., seconds	275-300°C. ^{2/} seconds	Percent	Gravity, °A.P.I.	Percent carbon residue
1...	Birch Run	58	87	175	12.8	17.5	7.4
2...	Saginaw	63	100	200	9.3	17.0	7.5
3...	Muskegon	55	75	125	25.4	17.0	8.2
4...	West Branch	61	91	170	29.6	16.0	9.4
5...	Edmore	56	75	125	14.8	21.5	3.7
6...	Denver Township	55	77	125	17.7	19.4	6.0
7...	Muskegon	53	75	120	24.8	16.2	9.0
8...	West Branch	55	80	135	33.6	13.3	13.8
9...	Crystal	56	75	115	19.1	21.1	4.3
10...	Porter	50	70	110	21.2	18.2	7.2
11...	Mount Pleasant (Greendale Twp.)	52	74	115	20.2	19.2	6.2
12...	Leaton	53	75	120	19.0	19.7	5.6
13...	Vernon	55	77	125	18.2	21.0	5.0
14...	Beaverton	55	77	125	19.1	18.1	7.1
15...	Monroe County	55	77	125	25.0	24.3	2.5
16...	Bradford (Pennsylvania)	60	81	130	19.7	26.6	1.7

^{2/} Key fraction 2.

Sample 1

Morse-Getty well no. 2
1525-1545 feet
Smith Petroleum Co.

Birch Run field
Berea sandstone

Michigan
Saginaw County
Sec. 20-10.N-6 E Mich. Merid.

GENERAL CHARACTERISTICS

Specific gravity, 0.814
Percent sulphur, 0.25
Saybolt Universal viscosity at 77°F., 38 seconds.
Saybolt Universal viscosity at 100°F., 36 seconds.

A.P.I. gravity, 42.3°
Pour point, below 5°F.
Color, dark green

DISTILLATION, BUREAU OF MINES HEMPEL METHOD

Dry distillation.

Barometer, 740 mm.

First drop: 36°C. (97°F.).

Temperature, °C	Per- cent cut	Sum per- cent	Sp.gr. of cut	°A.P.I. of cut	Viscosity at 100°F.	Cloud test, °F.	Temperature, °F.
Up to 50	-	-	-	-	-	-	Up to 122
50 - 75	4.3	4.3	0.650	86.2	-	-	122 - 167
75 - 100	5.7	10.0	.699	70.9	-	-	167 - 212
100 - 125	8.5	18.5	.726	63.4	-	-	212 - 257
125 - 150	7.2	25.7	.744	58.7	-	-	257 - 302
150 - 175	6.9	32.6	.763	54.0	-	-	302 - 347
175 - 200	6.0	38.6	.781	49.7	-	-	347 - 392
200 - 225	6.0	44.6	.796	46.3	-	-	392 - 437
225 - 250	6.7	51.3	.809	43.4	-	-	437 - 482
250 - 275	8.1	59.4	.823	40.4	-	-	482 - 527

Vacuum distillation at 40 mm

Up to 200	4.5	4.5	0.845	36.0	41	10	Up to 392
200 - 225	6.4	10.9	.854	34.2	46	25	392 - 437
225 - 250	5.1	16.0	.869	31.3	58	45	437 - 482
250 - 275	4.3	20.3	.884	28.6	87	60	482 - 527
275 - 300	5.4	25.7	.895	26.6	175	80	527 - 572

Carbon residue of residuum, 7.4 percent. Carbon residue of crude, 0.9 percent.

APPROXIMATE SUMMARY

	Percent	Sp.gr.	°A.P.I.	Viscosity
Light gasoline	10.0	0.678	77.2	-
Total gasoline and naphtha	38.6	0.732	61.3	-
Kerosene distillate	20.8	.811	43.0	-
Gas oil	9.7	.850	35.0	-
Nonviscous lubricating distillate .	9.2	.859-.885	33.2-28.4	50-100
Medium lubricating distillate	5.5	.885-.898	28.4-26.1	100-200
Viscous lubricating distillate	1.3	.898-.901	26.1-25.6	Above 200
Residuum	12.8	.950	17.5	-
Distillation loss	2.1	-	-	-

Sample 2

Voight well no. 5
1843-1858 feet
Sun Oil Co.

Saginaw field
Berea sandstone

Michigan
Saginaw County
Sec. 10-12 N-4 W. Mich. Merid.

GENERAL CHARACTERISTICS

Specific gravity, 0.805.

Percent sulphur, 0.24.

Saybolt Universal viscosity at 77°F., 35 seconds.

Saybolt Universal viscosity at 100°F., 34 seconds.

A.P.I. gravity, 44.3°.

Pour point, below 5°F.

Color, dark green.

DISTILLATION, BUREAU OF MINES HEMPEL METHOD

Dry distillation.			Barometer, 739 mm.		First drop: 36°C. (97°F.).		
Temperature, °C	Per- cent cut	Sum per- cent	Sp.gr. of cut	A.P.I. of cut	Viscosity at 100°F.	Cloud test, °F.	Temperature, °F.
Up to 50	-	-	-	-	-	-	Up to 122
50 - 75	3.6	3.6	0.658	83.6	-	-	122 - 167
75 - 100	6.8	10.4	.702	70.1	-	-	167 - 212
100 - 125	10.4	20.8	.726	63.4	-	-	212 - 257
125 - 150	8.6	29.4	.743	58.9	-	-	257 - 302
150 - 175	7.8	37.2	.763	54.0	-	-	302 - 347
175 - 200	6.8	44.0	.780	49.9	-	-	347 - 392
200 - 225	6.9	50.9	.796	46.3	-	-	392 - 437
225 - 250	6.8	57.7	.811	43.0	-	-	437 - 482
250 - 275	8.2	65.9	.824	40.2	-	-	482 - 527

Vacuum distillation at 40 mm

Up to 200	4.3	4.3	0.847	35.6	41	10	Up to 392
200 - 225	5.6	9.9	.857	33.6	47	25	392 - 437
225 - 250	4.9	14.8	.872	30.8	63	45	437 - 482
250 - 275	3.0	17.8	.883	27.9	100	60	482 - 527
275 - 300	4.7	22.5	.900	25.7	200	75	527 - 572

Carbon residue of residuum, 7.5 percent

Carbon residue of crude, 0.7 percent

APPROXIMATE SUMMARY

	Percent	Sp.gr.	A.P.I.	Viscosity
Light gasoline	10.4	0.687	74.5	
Total gasoline and naphtha	44.0	.735	61.0	--
Kerosene distillate	21.9	.811	43.0	--
Gas oil	8.2	.851	34.8	--
Nonviscous lubricating distillate ..	8.1	.860-.888	33.0-27.9	50-100
Medium lubricating distillate	3.9	.888-.900	27.9-25.7	100-200
Viscous lubricating distillate	2.3	.900-.907	25.7-24.5	Above 200
Residuum	9.3	.953	17.0	--
Distillation loss	2.3	-	-	--

Sample 3

Fred Figge well no. 1
1,661-1,667 feet
Muskegon Oil Corporation.

Muskegon field
Upper Traverse limestone

Michigan
Muskegon County
Sec. 9-10 N-16 W. Principal
Merid.

GENERAL CHARACTERISTICS

Specific gravity, 0.838.

Percent sulphur, 0.56.

Saybolt Universal viscosity at 77°F., 50 seconds.

Saybolt Universal viscosity at 100°F., 44 seconds.

A.P.I. gravity, 37.4°.

Pour point, below 5°F.

Color, greenish black.

DISTILLATION, BUREAU OF MINES HEMPEL METHOD

Dry distillation.

Barometer, 740 mm. First drop: 38°C. (100°F.).

Temperature, °C	Per- cent cut	Sum per- cent	Sp.gr. of cut	°A.P.I. of cut	Viscosity at 100°F.	Cloud test, °F.	Temperature, °F.
Up to 50	-	-	-	-	-	-	Up to 122
50 - 75	3.8	3.8	0.654	84.9	-	-	122 - 167
75 - 100	3.5	7.3	.700	70.6	-	-	167 - 212
100 - 125	4.5	11.8	.726	63.4	-	-	212 - 257
125 - 150	5.1	16.9	.746	58.2	-	-	257 - 302
150 - 175	5.2	22.1	.763	54.0	-	-	302 - 347
175 - 200	5.8	27.9	.777	50.6	-	-	347 - 392
200 - 225	6.0	33.9	.790	47.6	-	-	392 - 437
225 - 250	6.6	40.5	.805	44.3	-	-	437 - 482
250 - 275	7.3	47.8	.819	41.3	-	-	482 - 527

Vacuum distillation at 40 mm

Up to 200	3.7	3.7	0.843	36.4	40	15	Up to 392
200 - 225	5.5	9.2	.852	34.6	45	30	392 - 437
225 - 250	4.8	14.0	.866	31.9	55	45	437 - 482
250 - 275	5.7	19.7	.880	29.3	75	60	482 - 527
275 - 300	5.3	25.0	.890	27.5	125	75	527 - 572

Carbon residue of residuum, 8.2 percent.

Carbon residue of crude, 2.1 percent.

APPROXIMATE SUMMARY

	Percent	Sp.gr.	°A.P.I.	Viscosity
Light gasoline	7.3	0.676	77.8	-
Total gasoline and naphtha	27.9	.734	61.3	-
Kerosene distillate	19.9	.806	44.1	-
Gas oil	9.1	.849	35.2	-
Nonviscous lubricating distillate ..	10.5	.859-.885	33.2-28.4	50-100
Medium lubricating distillate	5.4	.885-.895	28.4-26.6	100-200
Viscous lubricating distillate	-	-	-	Above 200
Residuum	25.4	.953	17.0	-
Distillation loss	1.8	-	-	-

Sample 4

A. Reinhardt well no, 1
1,711-1,721 feet
Weber Oil Co.

West Branch field
Traverse limestone

Michigan
Ogemaw County
Sec. 35-22 N-2 E Mich. Merid.

GENERAL CHARACTERISTICS

Specific gravity, 0.855.

A.P.I. gravity, 34.0°.

Percent sulphur, 0.35.

Pour point, below 5° F.

Saybolt Universal viscosity at 77°F., 58 seconds.

Color, brownish black.

Saybolt Universal viscosity at 100°F., 52 seconds.

DISTILLATION, BUREAU OF MINES HEMPEL METHOD

Dry distillation.

Barometer, 739 mm. First drop: 50°C. (122°F.).

Temperature, °C	Per- cent cut	Sum per- cent	Sp.gr. of cut	°A.P.I. of cut	Viscosity at 100°F.	Cloud test, °F	Temperature, °F
Up to 50	-	-	-	-	-	-	Up to 122
50 - 75	2.0	2.0	0.668	80.3	-	-	122 - 167
75 - 100	4.5	6.5	.706	68.9	-	-	167 - 212
100 - 125	6.4	12.9	.732	61.8	-	-	212 - 257
125 - 150	5.5	18.4	.751	56.9	-	-	257 - 302
150 - 175	5.8	24.2	.773	51.6	-	-	302 - 347
175 - 200	3.8	28.0	.790	47.6	-	-	347 - 392
200 - 225	4.9	32.9	.804	44.5	-	-	392 - 437
225 - 250	5.1	38.0	.816	41.9	-	-	437 - 482
250 - 275	6.5	44.5	.830	39.0	-	-	482 - 527

Vacuum distillation at 40 mm

Up to 200	4.5	4.5	0.850	35.0	41	15	Up to 392
200 - 225	6.0	10.5	.861	32.8	47	35	392 - 437
225 - 250	4.4	14.9	.872	30.8	61	50	437 - 482
250 - 275	5.1	20.0	.884	28.6	91	65	482 - 527
275 - 300	5.7	25.7	.896	26.4	170	80	527 - 572

Carbon residue of residuum, 9.4 percent.

Carbon residue of crude, 2.8 percent.

APPROXIMATE SUMMARY

	Percent	Sp.gr.	°A.P.I.	Viscosity
Light gasoline	6.5	0.694	72.4	
Total gasoline and naphtha	28.0	.743	58.9	-
Kerosene distillate	10.0	.810	43.2	-
Gas oil	15.2	.844	36.2	-
Nonviscous lubricating distillate .	9.4	.863-.885	32.5-28.4	50-100
Medium lubricating distillate	6.8	.885-.901	28.4-25.6	100-200
Viscous lubricating distillate8	.901-.902	25.6-25.4	Above 200
Residuum	29.6	.959	16.0	-
Distillation loss2	-	-	-

Sample 5

H. P. Christensen well no. 5
3,025-3,097 feet
The Daily Crude Oil Co.

Edmore field
Traverse Limestone

Michigan
Montcalm County
Sec. 10-12 N-6 W Mich. Merid.

GENERAL CHARACTERISTICS

Specific gravity, 0.810.

Percent Sulphur, 0.11.

Saybolt Universal viscosity at 77°F., 37 seconds.

Saybolt Universal viscosity at 100°F., 35 seconds.

A.P.I. gravity, 43.20

Pour point, below 5°F.

Color, brownish green.

DISTILLATION, BUREAU OF MINES HEMPEL METHOD

Dry distillation.

Barometer, 743 mm.

First drop: 39°C. (102°F.).

Temperature, °C	Per- cent cut	Sum per- cent	Sp.gr. of cut	°A.P.I. of cut	Viscosity at 100°F.	Cloud test, °F.	Temperature, °F.
Up to 50	-	-	-	-	-	-	Up to 122
50 - 75	4.8	4.8	0.663	81.9	-	-	122 - 167
75 - 100	6.9	11.7	.706	68.9	-	-	167 - 212
100 - 125	8.8	20.5	.733	61.5	-	-	212 - 257
125 - 150	7.5	28.0	.753	56.4	-	-	257 - 302
150 - 175	7.3	35.3	.771	52.0	-	-	302 - 347
175 - 200	5.8	41.1	.787	48.3	-	-	347 - 392
200 - 225	6.1	47.2	.801	45.2	-	-	392 - 437
225 - 250	5.8	53.0	.815	42.1	-	-	437 - 482
250 - 275	6.7	59.7	.829	39.2	-	-	482 - 527

Vacuum distillation at 40 mm

Up to 200	3.5	3.5	0.848	35.4	40	15	Up to 392
200 - 225	5.7	9.2	.855	34.0	45	35	392 - 437
225 - 250	5.1	14.3	.864	32.3	56	55	437 - 482
250 - 275	4.7	19.0	.874	30.4	75	70	482 - 527
275 - 300	5.1	24.1	.883	28.8	125	90	527 - 572

Carbon residue of residuum, 3.7 percent. Carbon residue of crude, 0.5 percent.

APPROXIMATE SUMMARY

	Percent	Sp.gr.	°A.P.I.	Viscosity
Light gasoline	11.7	0.688	74.2	
Total gasoline and naphtha.....	41.1	.738	60.2	-
Kerosene distillate.....	11.9	.808	43.6	-
Gas oil.....	15.5	.842	36.6	-
Nonviscous lubricating distillate.	10.3	.859- .878	33.2- 29.7	50-100
Medium lubricating distillate.....	5.0	.878- .883	29.7- 27.9	100-200
Viscous lubricating distillate....	-	-	-	Above 200
Residuum.....	14.8	.925	21.5	-
Distillation loss.....	1.4	-	-	-

R. I. 3346

Sample 6

T. E. McCann well no. 1
2,975 - 3,104 feet
Brehm-Holman Oil Co.

Denver Township field
Traverse limestone

Michigan
Midland County
Sec. 20-15 N-2 W Mich. Merid.

GENERAL CHARACTERISTICS

Specific gravity, 0.807.

Percent sulphur, 0.12.

Saybolt Universal viscosity at 77°F., 38 seconds

Saybolt Universal viscosity at 100°F., 36 seconds.

A.P.I. gravity, 43.8°.

Pour point below 5°F.

Color, dark green.

DISTILLATION, BUREAU OF MINES HEMPEL METHOD

Dry distillation			Barometer, 743 mm.		First drop: 41°C. (106°F.).		
Temperature, °C.	Per- cent cut	Sum per- cent	Sp.gr. of cut	°A.P.I. of cut	Viscosity at 100°F.	Cloud test °F.	Temperature, °F.
Up to 50	-	-	-	-	-	-	Up to 122
50 - 75	7.1	7.1	0.653	85.2	-	-	122 - 167
75 - 100	5.3	12.4	.695	72.1	-	-	167 - 212
100 - 125	6.3	18.7	.722	64.5	-	-	212 - 257
125 - 150	5.6	24.3	.740	59.7	-	-	257 - 302
150 - 175	5.6	29.9	.756	55.7	-	-	302 - 347
175 - 200	6.2	36.1	.770	52.3	-	-	347 - 392
200 - 225	5.6	41.7	.784	49.0	-	-	392 - 437
225 - 250	6.3	48.0	.799	45.6	-	-	437 - 482
250 - 275	7.8	55.8	.811	43.0	-	-	482 - 527

Vacuum distillation at 40 mm

Up to 200	4.6	4.6	0.835	38.0	40	15	Up to 392
200 - 225	5.1	9.7	.846	35.8	43	30	392 - 437
225 - 250	4.9	14.6	.864	32.3	55	45	437 - 482
250 - 275	4.3	18.9	.878	29.7	77	60	482 - 527
275 - 300	4.6	23.5	.891	27.3	125	75	527 - 572

Carbon residue of residuum, 6.0 percent. Carbon residue of crude, 1.1 percent.

APPROXIMATE SUMMARY

	Percent	Sp.gr.	°A.P.I.	Viscosity
Light gasoline	12.4	0.671	79.4	-
Total gasoline and naphtha.....	36.1	.721	64.8	-
Kerosene distillate.....	19.7	.799	45.6	-
Gas oil.....	10.2	.844	36.2	-
Nonviscous lubricating distillate..	8.7	.857- .884	33.6- 28.6	50-100
Medium lubricating distillate.....	4.6	.884- .897	28.6- 26.3	100-200
Viscous lubricating distillate.....	-	-	-	Above 200
Residuum.....	17.7	.938	19.4	-
Distillation loss.....	3.0	-	-	-

Sample 7

C. E. Myler well no. 2
2,035-2,062 feet
Muskegon Development Co.

Muskegon field
Dundee limestone

Michigan
Muskegon County
Sec. 8-10 N-16 W Principal Merid.

GENERAL CHARACTERISTICS

Specific gravity, 0.845.
Percent sulphur, 1.03
Saybolt Universal viscosity at 77°F., 51 seconds.
Saybolt Universal viscosity at 100°F., 45 seconds.

A.P.I. gravity, 36.0°.
Pour point, below 5°F.
Color, brownish green.

DISTILLATION, BUREAU OF MINES HEMPEL METHOD

Dry distillation. Barometer, 741 mm. First drop: 66°C. (151°F.).

Temperature, °C.	Per- cent cut	Sum per- cent	Sp.gr. of cut	°A.P.I. of cut	Viscosity at 100°F.	Cloud test, °F.	Temperature, °F.
Up to 50	-	-	-	-	-	-	Up to 122
50 - 75	-	-	-	-	-	-	122 - 167
75 - 100	2.9	2.9	0.694	72.4	-	-	167 - 212
100 - 125	4.0	6.9	.721	64.8	-	-	212 - 257
125 - 150	5.2	12.1	.737	60.5	-	-	257 - 302
150 - 175	5.7	17.8	.751	56.9	-	-	302 - 347
175 - 200	8.2	26.0	.763	54.0	-	-	347 - 392
200 - 225	7.4	33.4	.776	50.9	-	-	392 - 437
225 - 250	7.7	41.1	.796	46.3	-	-	437 - 482
250 - 275	8.5	49.6	.812	42.8	-	-	482 - 527

Vacuum distillation at 40 mm

Up to 200	3.5	3.5	0.841	36.8	39	15	Up to 392
200 - 225	5.5	9.0	.853	34.4	43	30	392 - 437
225 - 250	5.0	14.0	.872	30.8	53	45	437 - 482
250 - 275	5.2	19.2	.888	27.9	75	60	482 - 527
275 - 300	6.2	25.4	.900	25.7	120	75	527 - 572

Carbon residue of residuum, 9.0 percent. Carbon residue of crude, 2.2 percent.

APPROXIMATE SUMMARY

	Percent	Sp.gr.	°A.P.I.	Viscosity
Light gasoline	2.9	0.694	72.4	
Total gasoline and naphtha	26.0	.741	59.5	-
Kerosene distillate	23.6	.795	46.5	-
Gas oil	9.9	.851	34.8	-
Nonviscous lubricating distillate ..	9.9	.866-.895	31.9-26.6	50-100
Medium lubricating distillate	5.6	.895-.906	26.6-24.7	100-200
Viscous lubricating distillate	-	-	-	Above 200
Residuum	24.8	.958	16.2	-
Distillation loss	0.2	-	-	-

R. I. 3346

Stella Wilcox well no. 2
2,499-2,677 feet
Weber Oil Co.

Sample 8
West Branch field
Dundee limestone

Michigan
Ogemaw County
Sec. 35-22 N-2 E Mich. Merid.

GENERAL CHARACTERISTICS

Specific gravity, 0.862.

Percent sulphur, 0.52.

Saybolt Universal viscosity at 77°F., 74 seconds.

Saybolt Universal viscosity at 100°F., 60 seconds.

A.P.I. gravity, 32.7°.

Pour point, below 5°F.

Color, brownish black.

DISTILLATION, BUREAU OF MINES HEMPEL METHOD

Dry distillation.

Barometer, 739 mm. First drop: 44°C. (111°F.).

Temperature, °C.	Per- cent cut	Sum per- cent	Sp.gr. of cut	°A.P.I. of cut	Viscosity at 100°F.	Cloud test, °F.	Temperature, °F
Up to 50	-	-	-	-	-	-	Up to 122
50 - 75	-	-	-	-	-	-	122 - 167
75 - 100	3.6	3.6	0.683	75.7	-	-	167 - 212
100 - 125	4.1	7.7	.720	65.0	-	-	212 - 257
125 - 150	4.5	12.2	.733	60.0	-	-	257 - 302
150 - 175	4.7	16.9	.754	56.2	-	-	302 - 347
175 - 200	5.1	22.0	.768	52.7	-	-	347 - 392
200 - 225	4.8	26.8	.781	49.7	-	-	392 - 437
225 - 250	5.8	32.6	.797	46.0	-	-	437 - 482
250 - 275	7.7	40.3	.809	43.4	-	-	482 - 527

Vacuum distillation at 40 mm

Up to 200	5.5	5.5	0.836	37.8	39	20	Up to 392
200 - 225	5.3	10.8	.848	35.4	44	35	392 - 437
225 - 250	4.8	15.6	.867	31.7	55	50	437 - 482
250 - 275	3.9	19.5	.885	28.4	80	65	482 - 527
275 - 300	5.2	24.7	.895	26.6	135	80	527 - 572

Carbon residue of residuum, 13.8 percent.

Carbon residue of crude, 4.6 percent.

APPROXIMATE SUMMARY

	Percent	Sp.gr.	°A.P.I.	Viscosity
Light gasoline	3.6	0.683	75.7	-
Total gasoline and naphtha	22.0	.736	60.8	-
Kerosene distillate	18.3	.798	45.8	-
Gas oil	10.9	.844	36.2	-
Nonviscous lubricating distillate ..	8.3	.858-.889	33.4-27.7	50-100
Medium lubricating distillate	5.5	.889-.900	27.7-25.7	100-200
Viscous lubricating distillate	-	-	-	Above 200.
Residuum	33.6	.977	13.3	-
Distillation loss	1.4	-	-	-

Sample 9

E. Jay Fulford well no. 1
3,216-3,221 feet
The Daily Crude Oil Co.

Crystal field
Dundee limestone

Michigan
Montcalm County
Sec. 36-11 N-5 W Mich. Merid.

GENERAL CHARACTERISTICS

Specific gravity, 0.819. A.P.I. gravity, 41.3°.
Percent sulphur, 0.37. Pour point, below 5°F.
Saybolt Universal viscosity at 77°F., 39 seconds. Color, brownish green.
Saybolt Universal viscosity at 100°F., 37 seconds.

DISTILLATION, BUREAU OF MINES HEMPEL METHOD

Dry distillation. Barometer, 739 mm. First drop: 37°C. (99°F.).

Temperature, °C.	Per- cent cut	Sum per- cent	Sp.gr. of cut	°A.P.I. of cut	Viscosity at 100°F.	Cloud test, °F.	Temperature, °F.
Up to 50	-	-	-	-	-	-	Up to 122
50 - 75	6.2	6.2	0.652	85.5	-	-	122 - 167
75 - 100	5.7	11.9	.695	72.1	-	-	167 - 212
100 - 125	6.4	18.3	.723	64.2	-	-	212 - 257
125 - 150	5.7	24.0	.741	59.5	-	-	257 - 302
150 - 175	5.9	29.9	.755	55.9	-	-	302 - 347
175 - 200	6.6	36.5	.766	53.2	-	-	347 - 392
200 - 225	5.7	42.2	.779	50.1	-	-	392 - 437
225 - 250	7.1	49.3	.794	46.7	-	-	437 - 482
250 - 275	8.2	57.5	.809	43.4	-	-	482 - 527

Vacuum distillation at 40 mm.

Up to 200	4.7	4.7	0.833	38.4	40	20	Up to 392
200 - 225	5.4	10.1	.842	36.6	44	35	392 - 437
225 - 250	4.9	15.0	.862	32.7	56	50	437 - 482
250 - 275	2.9	17.9	.871	31.0	75	65	482 - 527
275 - 300	5.3	23.2	.882	28.9	115	80	527 - 572

Carbon residue of residuum, 4.3 percent. Carbon residue of crude, 0.8 percent.

APPROXIMATE SUMMARY

	Percent	Sp.gr.	°A.P.I.	Viscosity
Light gasoline	11.9	0.673	73.8	
Total gasoline and naphtha	36.5	.722	64.5	-
Kerosene distillate	21.0	.796	46.3	-
Gas oil	10.1	.841	36.8	-
Nonviscous lubricating distillate ..	9.0	.852-.878	34.6-29.7	50-100
Medium lubricating distillate	4.1	.878-.889	29.7-27.7	100-200
Viscous lubricating distillate	-	-	-	Above 200
Residuum	19.1	.927	21.1	-
Distillation loss2	-	-	-

Sample 10

A. J. Megill well no. 1
3,365-3,436 feet
Fred Turner

Porter field
Dundee limestone

Michigan
Midland County

Sec. 28-13 N-1 W Mich. Merid.

GENERAL CHARACTERISTICS

Specific gravity, 0.822.

Percent sulphur, 0.26.

Saybolt Universal viscosity at 77°F., 41 seconds.

Saybolt Universal viscosity at 100°F., 38 seconds.

A.P.I. gravity, 40.6°.

Pour point, below 5°F.

Color, dark green.

DISTILLATION, BUREAU OF MINES HEMPEL METHOD

Dry distillation

Barometer, 740 mm. First drop: 37°C. (99°F.).

Temperature, °C.	Per- cent cut	Sum per- cent	Sp.gr. of cut	°A.P.I. of cut	Viscosity at 100°F.	Cloud test, °F	Temperature, °F
Up to 50	-	-	-	-	-	-	Up to 122
50 - 75	3.9	3.9	0.654	84.9	-	-	122 - 167
75 - 100	4.3	8.2	.689	73.9	-	-	167 - 212
100 - 125	5.5	13.7	.715	66.4	-	-	212 - 257
125 - 150	5.3	19.0	.733	61.5	-	-	257 - 302
150 - 175	5.5	24.5	.748	57.7	-	-	302 - 347
175 - 200	6.3	30.8	.761	54.4	-	-	347 - 392
200 - 225	6.8	37.6	.779	50.1	-	-	392 - 437
225 - 250	7.1	44.7	.795	46.5	-	-	437 - 482
250 - 275	8.1	52.8	.807	43.8	-	-	482 - 527

Vacuum distillation at 40 mm

Up to 200	4.7	4.7	0.831	38.8	38	15	Up to 392
200 - 225	5.8	10.5	.845	36.0	42	30	392 - 437
225 - 250	4.8	15.3	.863	32.5	50	45	437 - 482
250 - 275	4.6	19.9	.880	29.3	70	60	482 - 527
275 - 300	5.1	25.0	.892	27.1	110	75	527 - 572

Carbon residue of residuum, 7.2 percent. Carbon residue of crude, 1.5 percent.

APPROXIMATE SUMMARY

	Percent	Sp.gr.	°A.P.I.	Viscosity
Light gasoline	8.2	0.672	79.1	
Total gasoline and naphtha	30.8	.722	64.5	-
Kerosene distillate	22.0	.794	46.7	-
Gas oil	12.9	.844	36.2	-
Nonviscous lubricating distillate ..	8.3	.863-.889	32.5-27.7	50-100
Medium lubricating distillate	3.8	.889-.898	27.7-26.1	100-200
Viscous lubricating distillate	-	-	-	Above 200
Residuum	21.2	.945	18.2	-
Distillation loss	1.0	-	-	-

Sample 11

Adams well no. 1
3,506-3,567 feet
Talbot Oil Co.

Greendale Township field
Dundee Limestone

Michigan
Midland County
Sec. 10-14 N-2 W Mich. Merid.

GENERAL CHARACTERISTICS

Specific gravity, 0.818.

Percent sulphur, 0.13.

Saybolt Universal viscosity at 77°F., 42 seconds.

Saybolt Universal viscosity at 100°F., 39 seconds.

A.P.I. gravity, 41.5°.

Pour point, below 5°F.

Color, brownish green.

DISTILLATION, BUREAU OF MINES HEMPEL METHOD

Dry distillation.

Barometer, 744 mm

First drop: 36°C. (97°F.).

Temperature, °C.	Per- cent cut	Sum per- cent	Sp.gr. of cut	°A.P.I. of cut	Viscosity at 100°F.	Cloud test, °F.	Temperature, °F.
Up to 50	-	-	-	-	-	-	Up to 122
50 - 75	3.5	3.5	0.648	86.9	-	-	122 - 167
75 - 100	3.4	6.9	.683	75.7	-	-	167 - 212
100 - 125	5.5	12.4	.714	66.7	-	-	212 - 257
125 - 150	5.7	18.1	.732	61.8	-	-	257 - 302
150 - 175	5.5	23.6	.746	58.2	-	-	302 - 347
175 - 200	7.3	30.9	.761	54.4	-	-	347 - 392
200 - 225	6.4	37.3	.778	50.4	-	-	392 - 437
225 - 250	6.7	44.0	.793	46.9	-	-	437 - 482
250 - 275	8.6	52.6	.807	43.8	-	-	482 - 527

Vacuum distillation at 40 mm

Up to 200	5.3	5.3	0.832	38.6	39	20	Up to 392
200 - 225	5.6	10.9	.844	36.2	43	35	392 - 437
225 - 250	4.8	15.7	.861	32.8	52	50	437 - 482
250 - 275	4.5	20.2	.877	29.9	74	65	482 - 527
275 - 300	5.0	25.2	.890	27.5	115	80	527 - 572

Carbon residue of residuum, 6.2 percent. Carbon residue of crude, 1.3 percent.

APPROXIMATE SUMMARY

	Percent	Sp.gr.	°A.P.I.	Viscosity
Light gasoline	6.9	0.665	81.3	-
Total gasoline and naphtha.....	30.9	.723	64.2	-
Kerosene distillate.....	21.7	.794	46.7	-
Gas oil.....	12.2	.843	36.4	-
Nonviscous lubricating distillate	8.8	.857- .885	33.6- 28.4	50-100
Medium lubricating distillate....	4.2	.885- .897	28.4- 26.3	100-200
Viscous lubricating distillate...	-	-	-	Above 200
Residuum.....	20.2	.939	19.2	-
Distillation loss.....	2.0	-	-	-

Sample 12

Mahon well no. 1
3,604-3,662 feet
Gibson, Johnson and
T. K. Buzzard.

Leaton field
Dundee limestone

Michigan
Isabella County
Sec. 24-15 N-4 W Mich. Merid.

GENERAL CHARACTERISTICS

Specific gravity; 0.811.

Percent sulphur, 0.16.

Saybolt Universal viscosity at 77°F., 40 seconds.

Saybolt Universal viscosity at 100°F., 37 seconds.

A.P.I. gravity, 43.0°.

Pour point, below 5°F.

Color, green.

DISTILLATION, BUREAU OF MINES HEMPEL METHOD

Dry distillation.			Barometer, 744 mm		First drop: 37°C. (99°F.).		
Temperature, °C	Per- cent cut	Sum per- cent	Sp.gr. of cut	°A.P.I. of cut	Viscosity at 100°F.	Cloud test, °F	Temperature °F.
Up to 50	-	-	-	-	-	-	Up to 122
50 - 75	5.5	5.5	0.650	86.2	-	-	122 - 167
75 - 100	4.5	10.0	.693	72.7	-	-	167 - 212
100 - 125	5.9	15.9	.717	65.9	-	-	212 - 257
125 - 150	5.6	21.5	.734	61.3	-	-	257 - 302
150 - 175	5.9	27.4	.748	57.7	-	-	302 - 347
175 - 200	6.5	33.9	.763	54.0	-	-	347 - 392
200 - 225	5.9	39.8	.779	50.1	-	-	392 - 437
225 - 250	6.9	46.7	.795	46.5	-	-	437 - 482
250 - 275	8.4	55.1	.808	43.6	-	-	482 - 527

Vacuum distillation at 40 mm							
Up to 200	4.3	4.3	0.832	38.6	39	15	Up to 392
200 - 225	5.2	9.5	.843	36.4	43	30	392 - 437
225 - 250	5.0	14.5	.860	33.0	53	45	437 - 482
250 - 275	4.3	18.8	.875	30.2	75	60	482 - 527
275 - 300	5.2	24.0	.887	28.0	120	80	527 - 572

Carbon residue of residuum, 5.6 percent. Carbon residue of crude, 1.1 percent.

APPROXIMATE SUMMARY

	Percent	Sp.gr.	°A.P.I.	Viscosity
Light gasoline	10.0	0.669	80.0	-
Total gasoline and naphtha.....	33.9	.720	65.0	-
Kerosene distillate.....	21.2	.796	46.3	-
Gas oil.....	10.5	.841	36.8	-
Nonviscous lubricating distillate.	8.8	.855- .881	34.0- 29.1	50-100
Medium lubricating distillate.....	4.7	.881- .893	29.1- 27.0	100-200
Viscous lubricating distillate.....	-	-	-	Above 200
Residuum.....	19.0	.936	19.7	-
Distillation loss.....	1.9	-	-	-

Sample 13

Bowman well no. 1
3,720-3,724 feet
Nollem Oil Co.

Vernon field
Dundee limestone

Michigan
Isabella County
Sec. 27-16 N-4 W Mich. Merid.

GENERAL CHARACTERISTICS

Specific gravity, 0.806.
Percent sulphur, 0.30.
Saybolt Universal viscosity at 77°F., 39 seconds.
Saybolt Universal viscosity at 100°F., 39 seconds.

A.P.I. gravity, 44.1°.
Pour point, below 5°F.
Color, brownish green.

DISTILLATION, BUREAU OF MINES HEMPEL METHOD

Dry distillation.			Barometer, 743 mm		First drop: 40°C. (104°F.).		
Temperature, °C	Per cent cut	Sum per- cent	Sp.gr. of cut	°A.P.I. of cut	Viscosity at 100°F.	Cloud test, °F	Temperature, °F
Up to 50	-	-	-	-	-	-	Up to 122
50 - 75	5.7	5.7	0.648	86.9	-	-	122 - 167
75 - 100	4.8	10.5	.689	73.9	-	-	167 - 212
100 - 125	6.2	16.7	.716	66.1	-	-	212 - 257
125 - 150	6.1	22.8	.734	61.3	-	-	257 - 302
150 - 175	5.9	28.7	.749	57.4	-	-	302 - 347
175 - 200	6.2	34.9	.762	54.2	-	-	347 - 392
200 - 225	6.3	41.2	.776	50.9	-	-	392 - 437
225 - 250	6.5	47.7	.791	47.4	-	-	437 - 482
250 - 275	8.8	56.5	.807	43.8	-	-	482 - 527

Vacuum distillation at 40 mm

Up to 200	4.2	4.2	0.833	38.4	39	15	Up to 392
200 - 225	5.7	9.9	.843	36.4	43	30	392 - 437
225 - 250	4.5	14.4	.859	33.2	55	45	437 - 482
250 - 275	4.3	18.7	.873	30.6	77	60	482 - 527
275 - 300	4.9	23.6	.885	28.4	125	75	527 - 572

Carbon residue of residuum, 5.0 percent. Carbon residue of crude, 0.9 percent.

APPROXIMATE SUMMARY

	Percent	Sp.gr.	°A.P.I.	Viscosity
Light gasoline	10.5	0.667	80.6	-
Total gasoline and naphtha.....	34.9	.718	65.6	-
Kerosene distillate.....	21.6	.793	46.9	-
Gas oil.....	10.1	.841	36.8	-
Nonviscous lubricating distillate..	8.7	.852- .879	34.6- 29.5	50-100
Medium lubricating distillate.....	4.8	.879- .891	29.5- 27.3	100-200
Viscous lubricating distillate.....	-	-	-	Above 200
Residuum.....	18.2	.928	21.0	-
Distillation loss.....	1.7	-	-	-

Sample 14

Campbell well no. 1
3,837-3,908 feet
Great Lakes Oil Co.

Beaverton field
Dundee limestone

Michigan
Gladwin County
Sec. 3-17 N-2 W Mich. Merid.

GENERAL CHARACTERISTICS

Specific gravity, 0.819.

Percent sulphur, 0.16.

Saybolt Universal viscosity at 77°F., 41 seconds.

Saybolt Universal viscosity at 100°F., 38 seconds.

A.P.I. gravity, 41.3°.

Pour point, below 5°F.

Color, brownish green.

DISTILLATION, BUREAU OF MINES HEMPEL METHOD

Dry distillation.

Barometer, 744 mm. First drop: 35°C. (95°F.).

Temperature, °C.	Per- cent cut	Sum per- cent	Sp.gr. of cut	A.P.I. of cut	Viscosity at 100°F.	Cloud test, °F.	Temperature, °F.
Up to 50	-	-	-	-	-	-	Up to 122
50 - 75	3.9	3.9	0.652	85.5	-	-	122 - 167
75 - 100	4.5	8.4	.697	71.5	-	-	167 - 212
100 - 125	6.2	14.6	.718	65.6	-	-	212 - 257
125 - 150	5.5	20.1	.734	61.3	-	-	257 - 302
150 - 175	6.2	26.3	.748	57.7	-	-	302 - 347
175 - 200	6.4	32.7	.763	54.0	-	-	347 - 392
200 - 225	6.4	39.1	.779	50.1	-	-	392 - 437
225 - 250	6.5	45.6	.794	46.7	-	-	437 - 482
250 - 275	8.8	54.4	.810	43.2	-	-	482 - 527

Vacuum distillation at 40 mm

Up to 200	4.9	4.9	0.835	38.0	40	20	Up to 392
200 - 225	5.9	10.8	.847	35.6	43	35	392 - 437
225 - 250	4.6	15.4	.866	31.9	55	50	437 - 482
250 - 275	4.4	19.8	.881	29.1	77	65	482 - 527
275 - 300	5.0	24.8	.894	26.8	125	80	527 - 572

Carbon residue of residuum, 7.1 percent.

Carbon residue of crude, 1.4 percent.

APPROXIMATE SUMMARY

	Percent	Sp.gr..	A.P.I.	Viscosity
Light gasoline	8.4	0.676	77.8	
Total gasoline and naphtha	32.7	.724	63.9	-
Kerosene distillate	21.7	.796	46.3	-
Gas oil	10.9	.844	36.2	-
Nonviscous lubricating distillate ..	8.9	.858-.887	33.4-28.0	50-100
Medium lubricating distillate	5.0	.887-.901	28.0-25.6	100-200
Viscous lubricating distillate	-	-	-	Above 200
Residuum	19.1	.946	18.1	-
Distillation loss	1.7	-	-	-

Sample 15

Roe well no. 1
2,072-2,108 feet
A. V. Oil & Gas Co.

Monroe County field
Trenton limestone

Michigan
Monroe County
Sec. 19-6 S-6 E Mich. Merid.

GENERAL CHARACTERISTICS

Specific gravity, 0.815.

Percent sulphur, 0.18.

Saybolt Universal viscosity at 77°F., 47 seconds

Saybolt Universal viscosity at 100°F., 43 seconds

A.P.I. gravity, 42.1°.

Pour point, below 5°F.

Color, green.

DISTILLATION, BUREAU OF MINES HEMPEL METHOD

Dry distillation.

Barometer, 741 mm. First drop: 39°C. (102°F.).

Temperature, °C	Per- cent cut	Sum per- cent	Sp.gr. of cut	°A.P.I. of cut	Viscosity at 100°F.	Cloud test, °F.	Temperature, °F.
Up to 50	-	-	-	-	-	-	Up to 122
50 - 75	4.7	4.7	0.654	84.9	-	-	122 - 167
75 - 100	4.0	8.7	.690	73.6	-	-	167 - 212
100 - 125	5.0	13.7	.715	66.4	-	-	212 - 257
125 - 150	5.0	18.7	.733	61.5	-	-	257 - 302
150 - 175	5.3	24.0	.749	57.4	-	-	302 - 347
175 - 200	4.9	28.9	.764	53.7	-	-	347 - 392
200 - 225	5.3	34.2	.777	50.6	-	-	392 - 437
225 - 250	5.5	39.7	.791	47.4	-	-	437 - 482
250 - 275	7.4	47.1	.803	44.7	-	-	482 - 527

Vacuum distillation at 40 mm

Up to 200	3.7	3.7	0.828	39.4	39	15	Up to 392
200 - 225	6.3	10.0	.837	37.6	44	35	392 - 437
225 - 250	5.3	15.3	.848	35.4	55	55	437 - 482
250 - 275	4.6	19.9	.858	33.4	77	70	482 - 527
275 - 300	6.3	26.2	.870	31.1	125	90	527 - 572

Carbon residue of residuum, 2.5 percent. Carbon residue of crude, 0.6 percent.

APPROXIMATE SUMMARY

	Percent	Sp.gr.	°A.P.I.	Viscosity
Light gasoline	8.7	0.671	79.4	-
Total gasoline and naphtha	28.9	.719	65.3	-
Kerosene distillate	18.2	.792	47.2	-
Gas oil	10.1	.834	38.2	-
Nonviscous lubricating distillate ..	10.2	.843-.864	36.4-32.3	50-100
Medium lubricating distillate	5.9	.864-.877	32.3-29.9	100-200
Viscous lubricating distillate	-	-	-	Above 200
Residuum	25.0	.908	24.3	-
Distillation loss	1.7	-	-	-

Sample 16

Artley well no. 71
1,771-1,881 feet
Sloan and Zook

Bradford field
Bradford third sand

Pennsylvania
McKean County

GENERAL CHARACTERISTICS

Specific gravity, 0.801
Percent sulphur, less than 0.1
Saybolt Universal viscosity at 100° F., 39 seconds

A.P.I. gravity, 45.20
Pour point, below 50° F.
Color, No. 6+

DISTILLATION, BUREAU OF MINES HEMPEL METHOD

Dry distillation,		Barometer, 744 mm			First drop: 34° C. (93° F.)		
Temperature, °C.	Per- cent cut	Sum per- cent	Sp.gr. of cut	°A.P.I. of cut	Viscosity at 100° F.	Cloud test, °F.	Temperature, °F.
Up to 50	3.6	3.6	0.662	82.2	---	---	Up to 122
50 - 75	3.6	7.2	.663	81.9	---	---	122 - 167
75 - 100	5.7	12.9	.700	70.6	---	---	167 - 212
100 - 125	7.6	20.5	.731	62.1	---	---	212 - 257
125 - 150	6.1	26.6	.752	56.7	---	---	257 - 302
150 - 175	5.6	32.2	.767	53.0	---	---	302 - 347
175 - 200	5.4	37.6	.777	50.6	---	---	347 - 392
200 - 225	5.5	43.1	.788	48.1	---	---	392 - 437
225 - 250	5.7	48.8	.802	44.9	---	---	437 - 482
250 - 275	6.3	55.1	.814	42.3	---	---	482 - 527

Vacuum distillation at 40 mm

Up to 200	3.6	3.6	0.832	38.6	41	20	Up to 392
200 - 225	5.8	9.4	.839	37.2	46	40	392 - 437
225 - 250	5.4	14.8	.846	35.8	60	60	437 - 482
250 - 275	3.8	18.6	.855	34.0	81	80	482 - 527
275 - 300	5.0	23.6	.864	32.3	130	90	527 - 572

Carbon residue of residuum, 1.7 percent.

Carbon residue of crude, 0.3 percent

APPROXIMATE SUMMARY

	Percent	Sp.gr.	°A.P.I.	Viscosity
Light gasoline	12.9	0.679	76.9	---
Total gasoline and naphtha	37.6	.729	62.6	---
Kerosene distillate	17.5	.802	44.9	---
Gas oil	8.2	.835	38.0	---
Nonviscous lubricating distillate	10.2	.841-.858	36.8-33.4	50-100
Medium lubricating distillate ...	5.2	.858-.869	33.4-31.3	100-200
Viscous lubricating distillate...	---	---	---	Above 200
Residuum	19.7	.895	26.6	---
Distillation loss	1.6	---	---	---

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UNITED STATES BUREAU OF MINES
JOHN W. FINCH, DIRECTOR

REPORT OF INVESTIGATIONS

THE IMPORTANCE OF ADEQUATE VOLTAGE
FOR DISTRIBUTION SYSTEMS
IN COAL MINES



BY

E. J. GLEIM

REPORT OF INVESTIGATIONS

DEPARTMENT OF THE INTERIOR - BUREAU OF MINES

IMPORTANCE OF ADEQUATE VOLTAGE FOR DISTRIBUTION
SYSTEMS IN COAL MINES^{1/}

By E. J. Gleim^{2/}

INTRODUCTION

The importance of "good voltage," especially for direct-current systems in coal mines, is not always fully appreciated. From the practical aspect there are three viewpoints from which low voltage may be considered, namely: (a) its effect upon operation of electrical equipment, (b) its effect upon cost of maintenance and power, and (c) its effect upon safety in the mine.

The design of electrical motor-driven equipment usually is based upon a given or assumed constant value for the voltage of the system; consequently, when there is an appreciable drop below this value, unsatisfactory operation is likely to follow. In the case of shaker conveyors, for example, one manufacturer has pointed out that these conveyors must be operated at a certain definite speed if maximum output is to be secured. As shunt motors are used in the conveyor drives, either a rise or fall in voltage will produce corresponding changes in speed and thus seriously affect the efficiency of the conveyor. The introduction of automatic control for mine equipment has been greatly retarded because the relays, contactors, and other special devices required have failed to function properly when attempt was made to operate them on systems subject to variable and low-voltage conditions.

Some of the immediate effects of low voltage are not serious, from the standpoint of hazard produced. For example, push-button-controlled starting equipment employs magnetic contactors that require a certain potential to close them and cut out steps of starting resistance. If the voltage is too low, the accelerating contactors will not function and the starting resistor will be burned out through remaining in circuit longer than intended. Thus, while no personal hazard may attend this result, the entire machine becomes inoperative.

^{1/} The Bureau of Mines will welcome reprinting of this paper, provided the following footnote acknowledgment is used: "Reprinted from U. S. Bureau of Mines Report of Investigations 3347."

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When electrical energy is wasted as heat because of inadequate feeder capacity, the cost of this wasted energy may assume proportions that cannot be ignored, especially if the power is purchased. Data secured during the operation of his coal-cutting machine proved conclusively to a manufacturer of mining machinery that low voltage results in unnecessary waste of energy. His machine was rated 30 horsepower at 250 volts and took 105 amperes at that rating. During the 18 minutes required to cut a place in a certain mine, the average voltage at the motor terminals of this machine was 183.2, or 73 percent of normal. The average horsepower delivered by the motor while cutting was 28. The average current taken during this period, however, was 146.6, or about 40 percent above the normal required for the full load of 30 horsepower; but, on the basis of the average of 28 horsepower, the average current was 46 percent in excess of that required for the cutting of the particular place. Thus, while the motor was delivering less than full load at the lower voltage, it was actually being subjected to double the heating effect, because the latter varies as the square of the current. In addition to the possibility of damage to the motor, the cost of this excess current would amount to a considerable sum in a year if the same low-voltage condition prevailed day in and day out. In any mine where loading machines, drills, locomotives, pumps, and other equipment are also taking power from the same circuits as the mining machines, it can readily be seen that the cost of wasted energy due to low voltage is likely to be multiplied proportionately.

On the other hand, the effects of low voltage can assume more serious aspects, especially in gassy and dusty mines. Abuse of equipment, particularly that which is started and stopped frequently, is almost certain to follow the drop in potential. As the voltage decreases it becomes increasingly difficult to start such equipment as locomotives and mining machines. Motormen and machine runners in these instances find that their machines will not accelerate as quickly as they formerly did under normal voltage and, consequently, these men operate their controllers more quickly between the starting and running positions. Thus, the starting current will be made to exceed safe values and, then, if fuses and circuit breakers fail to protect the circuit, heating of brushes, armatures, and field coils will take place, making satisfactory maintenance difficult and costly.

Often when the motor fails to develop enough torque to start the load, the controller will be left in the "on" position for sustained periods, resulting in further heating, to say nothing of the damage to the controller fingers in making and breaking the heavy currents. If this is continued, the insulation is roasted and ultimately fails and then short circuits and grounds occur. Short circuits sometimes cause holes to be burned through housings of controllers, rheostats, and other electrical parts. Should this take place in a gassy atmosphere, ignition would very likely follow. In case the circuit becomes grounded, persons touching the metal parts of the machine may be subjected to shock.

On account of space limitations, trailing cables are generally of a smaller size than should be used for maximum efficiency and, consequently, heating frequently occurs. However, when the voltage is below normal, the cable may become excessively heated and its life materially shortened, or it may even take fire, particularly when there are poorly made splices in it.

Some Causes for Low Voltage

When the load is increased beyond the proper carrying capacity of the circuits, or when the resistance of the circuit is permitted to increase, then low voltage conditions may be expected to exist, at least at points distant from the generating station. It sometimes happens that the men who operate mining machines and locomotives complain of "bad power" in a mine where conditions were once entirely satisfactory. Search does not have to be carried very far to disclose that it is not the capacity of the generating equipment that has decreased but, rather, that the circuits have been extended as the mine developed and without increasing the size of conductors as the lines were lengthened. It has simply meant failure to realize that, as the electrical load moved farther and farther away from the generating station, the resistance of the circuits was being increased until the voltage drop became appreciable and caused trouble.

On the other hand, it not infrequently occurs that, although the positive side of the circuits is maintained at adequate capacity and low resistance, it is not realized that the negative is an equally important side of the circuit and therefore must have adequate capacity and low resistance. As a consequence of this, the bonding of rail joints has been neglected where the track is used as the negative or return side of the circuit in the direct-current systems commonly used in bituminous coal mines.

Low voltage is also caused by loading the generating equipment beyond its capacity, and at mines having steam-engine drives for the generators, the voltage has been lowered when boiler trouble caused the speed of the engines to drop with the drop in steam pressure.

Value of a Voltage Survey

A voltage survey made with recording instruments furnishes valuable information regarding the condition of the mine distribution system. Although indicating instruments may serve to give instantaneous readings that are useful in judging conditions at any particular moment, voltage charts taken simultaneously at several points are far superior in that they give a permanent record for study and analysis in determining whether steps should be taken to improve conditions.

Such an important matter as that of maintaining good voltage may be partially or wholly lost sight of in the strenuous efforts to produce the required daily tonnage of coal, until the failure of equipment brings about the realization of needed attention. Therefore, a careful periodic analysis

of conditions is essential if safe, efficient, and economical operation is desired, even as adequacy of ventilation must be checked from time to time with anemometers.

When conditions indicate that the voltage is lower than it should be, recording voltmeters should be connected to the system at points at which it is desired to secure the necessary data. Records should be obtained simultaneously at several points in the mine, if possible, and for the entire working shift, or, preferably, for 24 hours. A study of these records, together with a knowledge of connected load, capacity of feeders, generating equipment, etc., will then show where improvement is most needed. From this study it should be possible to determine what remedy to apply. It may be that more of the voltage drop occurs in the return circuit than the size of rail warrants. Measurements of the resistance of rail joints will then serve to indicate whether the bonding has been properly done. An inexpensive bond-testing instrument should be available for every mine electrician to use for this purpose.

BUREAU OF MINES' VOLTAGE SURVEYS

A study of voltage conditions underground was made by engineers of the Bureau of Mines with a threefold purpose in view, namely: (1) to obtain first-hand data by means of which to substantiate claims of machine manufacturers and the Bureau that voltage conditions are often so poor as to render the use of permissible equipment hazardous, (2) to stimulate general interest in better voltage conditions, and (3), if possible, to secure evidence of improved voltage conditions by means of charts made both before and after certain changes were made in the distribution system.

Equipment Used

In order to secure the desired data, four portable recording voltmeters were purchased. These had 24-hour circular paper charts driven by a spring clock movement that would run at least 24 hours. Some of the charts had 300-volt scales and some had 600-volt scales. The meters also had 300- and 600-volt terminals. This permitted their use on either 250- or 500-volt direct-current systems. Special "boxes" or frames made of wood were used for convenience in leveling and mounting the meters in a suitable position. Usually, the frame was fastened to a prop, the wires to the meter being brought through the back of the frame.

The movement of the needle was not damped in these meters as originally furnished. It was soon learned that the voltage often fluctuated too rapidly to give a satisfactory chart record and, accordingly, a damping device was added to each meter to prevent overtravel of the needle during sudden changes in voltage.

Procedure

The usual practice in securing voltage records was to connect one meter to the bus bars on the switchboard in the power house or substation.

The remaining three meters were then taken into the mine and connected at such points as were known or suspected of having low voltage because of excessive load, distance from generating station, inadequate feeder capacity, or other reasons. Usually, the meters were connected between the trolley wire and rail and allowed to remain connected until the one or more desired 24-hour charts had been secured. When records were to be secured at more than three places in the mine, the necessary meters were moved to the other places after enough records had been obtained in their original positions. Meanwhile, the chart on the meter in the power house or substation was changed each day. The total or maximum drop in voltage was then determined by a comparison of a chart in the mine with one secured for the switchboard voltage.

This procedure was followed in nine mines, which, for convenience of reference, will be designated as mines A, B, C, D, E, etc. Five of these mines were in western Pennsylvania, three were in West Virginia, and one was in Alabama. At one of these mines the nominal voltage was 500 and at the rest the voltage was 250.

RESULTS

Mine A

The first mine surveyed for voltage conditions had a 250-volt direct-current system in which the power supply was derived from two steam-engine-driven generators of 200 kw. capacity each, and a substation with a 200 kw. motor-generator set. The main power station was near the hoisting shaft and the substation was about 2 miles distant on the surface. Five points in the mine were selected for taking voltage charts, and 4 months later, after some changes had been made in the distribution system, charts were again taken at three of the same points. During the interval between the two sets of charts, steps had been taken to improve voltage conditions underground, as follows:

The "return" in some entries was assisted by the use of 4/0 wire along the floor, and in one of these entries some overhead 4/0 trolley wire (positive) was replaced by wire of 6/0 size.

The voltage maintained at the switchboard in the power plant was 275, with ordinary fluctuations running about 10 volts above or below this value. When the meters were first installed, the one that was farthest from the air shaft, through which feeder cables from the power plant entered the mine, recorded voltages of 150 or less for periods of 1/2 hour. The distance from the bottom of this air shaft to the meter was about 7,259 feet. After the improvements in the distribution system were completed, the voltmeter charts made at this distant point indicated a slightly higher average voltage, although there were still short periods when the voltage sank to 150. This represented a drop of about 110 volts or 42 percent.

Mine B

The power supply at this mine consisted of two 200 kw., 275-volt motor-generator sets in an outside substation not far from the drift mouth, and also a 200 kw. motor-generator set placed over part of the active mine workings. Connection of the first substation to the direct-current trolley system was made at two points - (1) outside the mine near the substation and (2) inside the mine in an active section adjacent to that served by the other substation. This latter connection was made by running feeders overland and dropping them down a borehole. Connection of the second substation to the system was also made by means of feeders carried down a borehole.

Seven places were chosen for installing the recording voltmeters at this mine. One was placed in the smaller substation, one was placed at a fan on the surface at the top of a slope, and the remaining places were underground.

Although the charts showed voltage fluctuations of considerable magnitude, the lowest values for periods of 5 minutes or more were 215 volts. These low values would be equivalent to a drop of approximately 55 volts, or 20 percent.

Most of the charts showed voltages averaging well over 250 for other parts of the mine. The lowest values and greatest fluctuations were recorded in a place in which the track comprising the "return" was not bonded on both sides. Furthermore, the place was about 2,900 feet from the nearest point where a feeder entered the mine, while the other meters were much closer than this.

Mine C

Mine C was another in which the direct-current distribution system was nominally for 250 volts.

This system, shown in diagram of figure 1, was fed from motor-generator sets at two points, namely: At A, by a feeder in the supply shaft near the engine room, and at B, by a feeder in the air shaft, which is about 9,600 feet away from the supply shaft. Six places in the mine, as indicated on the diagram, were chosen for the installation of the recording voltmeters. One meter was also connected to the bus bars in the engine room and remained there throughout the time when the other meters were in the mine. The charts taken in the engine room showed that a potential of 265 to 275 volts was maintained in the substation at this point. Inside the mine, voltage readings averaged from 225 to 255. This would correspond to a maximum drop of 50 volts, or 18 percent. However, there were short intervals in which the voltage was down to 100, or more than a 60 percent drop. One point, C, where this low value was recorded was about 5,900 feet from the nearest substation and about 14,800 feet from the substation in the engine room. Figure 2 is the chart that was taken at that point. The circuit in this particular section of the mine consisted of 4/0 trolley wire with 25-pound rail for the return.

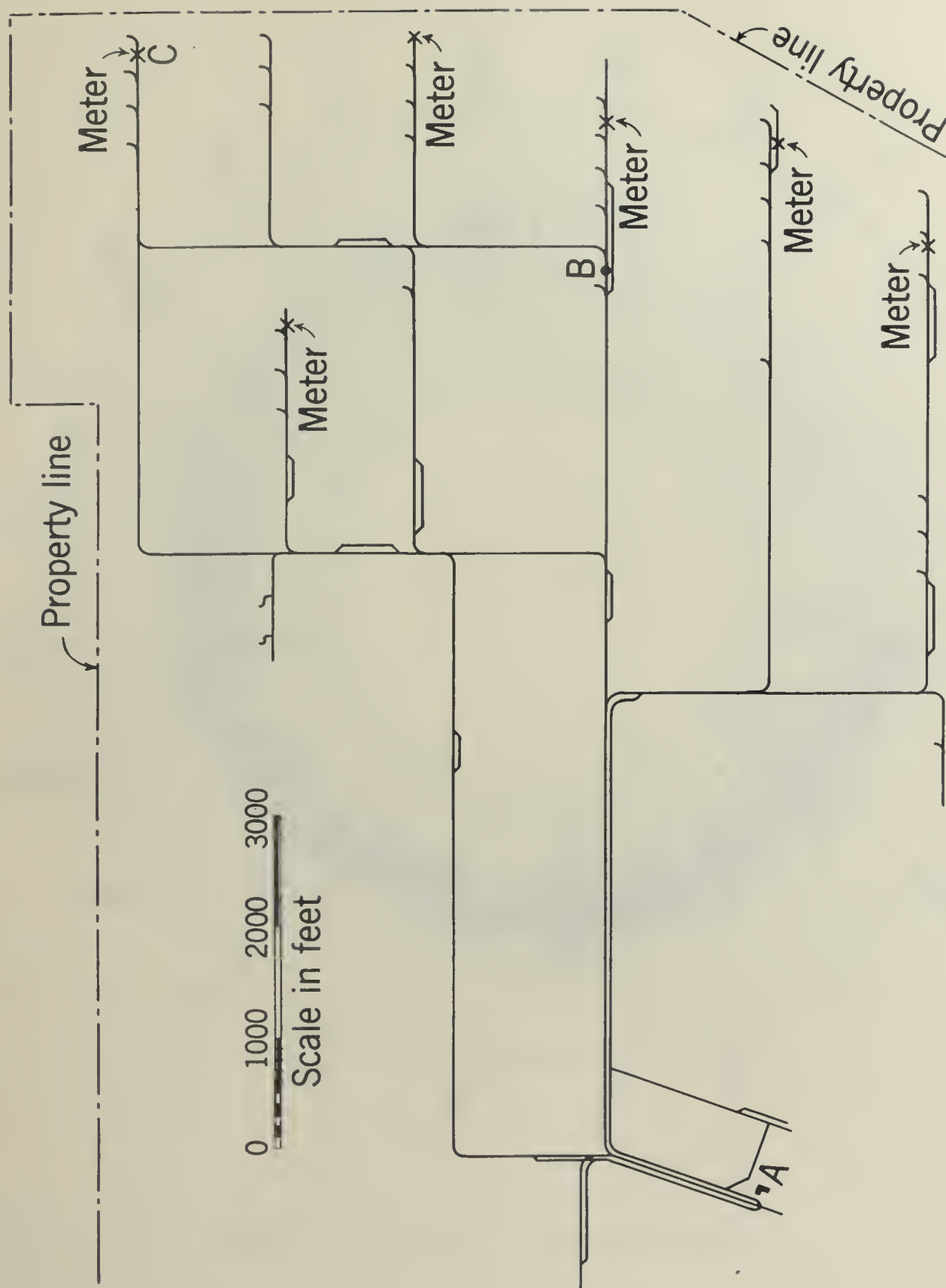


Figure 1.— Distribution system, mine C.

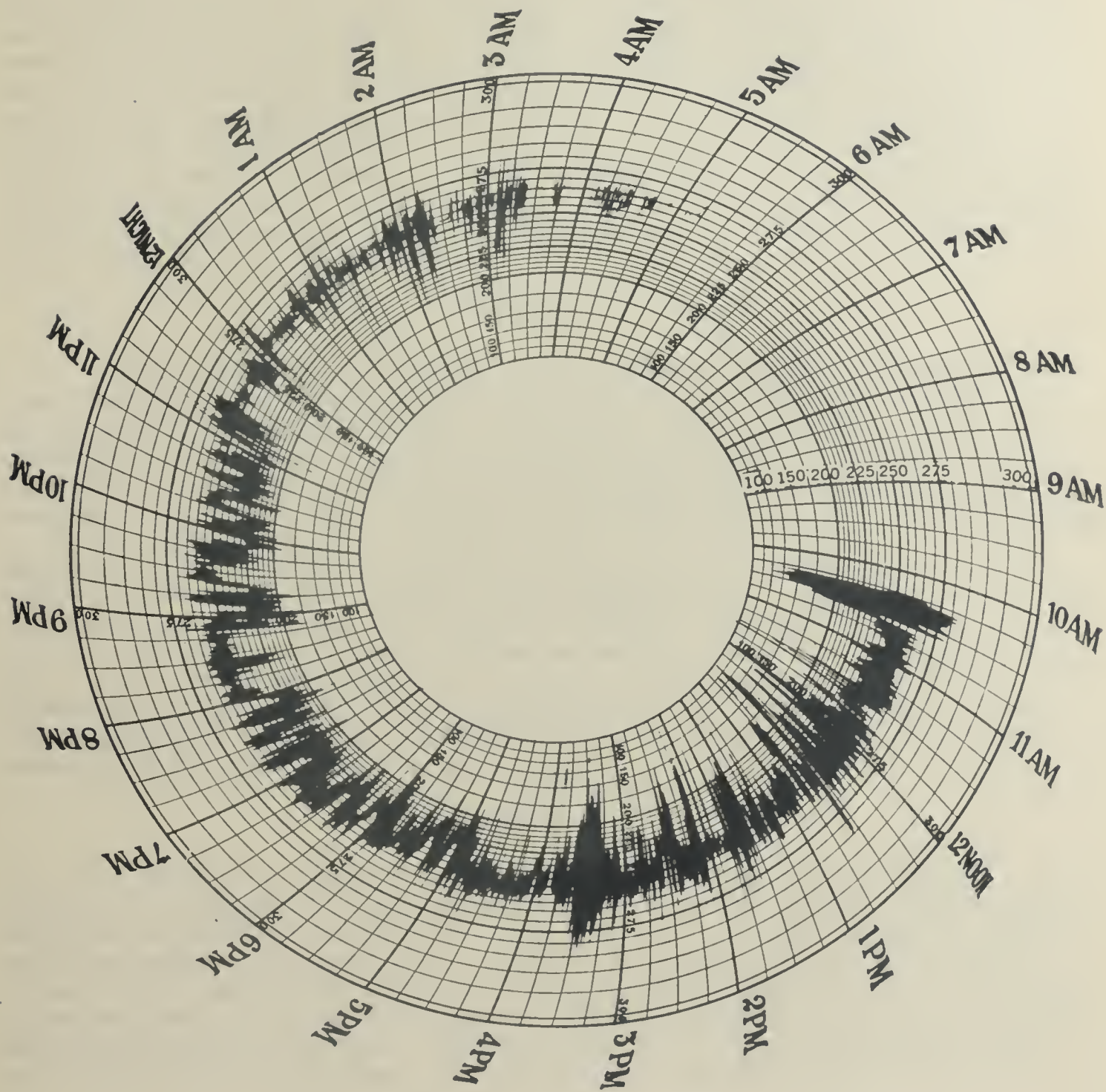


Figure 2.- Typical voltage chart, mine C.

Mine D

The distribution system in mine D was for 250 volts direct current. The power supply was two 200-kw. motor-generator sets in a substation near the shaft down which the feeder cables were taken. One voltmeter was placed in the substation to record the voltage fluctuations at that point, and meters were also set up at four places in the mine. The substation charts showed that a voltage of about 285 was maintained at the switchboard the greater part of the time, although there were variations from 275 to 290. The charts taken in the mine showed rapid fluctuations in the voltage, which ranged from about 200 to 285. The lowest sustained voltage was approximately 200, or 80 volts less than that in the substation. This meant a drop of about 28 percent. On several occasions this value was recorded for an appreciable time, and once it remained there for 5 minutes.

Mine E

Mine E had two substations on the surface and four underground. These were rated at 275 volts and totaled 750 kw. in capacity. The mine workings were somewhat scattered, but the most active section was selected for obtaining voltage data. For this purpose one voltmeter was installed in the substation nearest the center of the active section, and meters were placed near the end of the trolley wire in three working places. The recorded voltage in the substation showed a regulation that was not very uniform, the maximum variation being 80 volts; in other words, the voltage dropped below 200 at times. Low values were recorded in the working places, also. On one occasion the voltage in two of the working places remained at about 150 for 10 to 15 minutes. At the same time the voltage at the substation was between 200 and 225. This 50-volt difference, or 25-percent drop, between the substation and working place occurred in spite of the fact that the two were not more than about 1,100 feet apart and the substation was interconnected with the other substations by 500,000 circular mil feeders.

Mine F

Mine F was the only one of those surveyed in which a 500-volt system was employed. The mine system was fed from two outside substations - one near the drift-mouth entrance and one fairly centrally located over the active workings. The first had a total rated capacity of 600 kw. and the other a total capacity of 1,200 kw. A meter was placed in the latter station to record the voltage at that point and meters were also placed at two widely separated points in the mine. The first was about 4,700 feet distant from this substation and the second was about 5,880 feet from it.

The charts taken in the substation showed that a voltage of 560 to 580 was maintained during the period of the survey. The charts in the mine showed rapid fluctuations during most of the time when the meters were connected. While the average voltage remained comparatively high, there were occasions when the voltage fell momentarily to 300, a drop of at least 46 percent. In one section of the mine there was a period of 15 minutes during

which the average voltage was about 470, corresponding to a 16-percent drop.

Mine G

This mine was one in which voltage readings were taken on two different occasions; that is, both before and after changes had been made to effect an improvement in the regulation of the 250-volt direct-current distribution system. At the beginning of the survey the direct-current requirements in the mine were supplied by two substations, one containing two 300 kw. synchronous converters and the other two 200 kw. synchronous converters. These were outside substations, and connections to the system were made by means of feeder cables in boreholes.

The changes made to secure better voltage conditions were as follows: (1) The power company raised the voltage to the substations from 2,300 to 4,000 to reduce the drop in the transmission lines; (2) the transformer secondary connections were changed to increase the voltage impressed on the synchronous converters, which, in turn, raised the direct current voltage; (3) another outside substation containing a synchronous converter of 500 kw. capacity was placed nearer the center of the active workings and connected to the system by cables suspended in a borehole; and (4) 2,000 feet of 1,000,000 circular mil feeder was added to extend the existing feeder. Trolley wire of 6/0 size and rail of 60-pound size also were extended about 3,000 feet in the same entry to replace smaller wire and rail.

The recording voltmeters were installed at six different places in the mine and in the substation containing the two 300-kw. converters before the foregoing changes were made and again when the changes were completed.

Before the changes in the distribution system were made, the voltage in the substation was fairly constant at about 295 volts. After those changes, however, the voltmeter needle was off scale most of the time, that is, the voltage was above the 300-volt limit of the scale. Readings with an indicating voltmeter gave values as high as 320 volts.

Figure 3 is a chart showing voltage conditions at the meter station farthest from the substation, a distance of 8,600 feet. This was before any changes in the system had been made. It will be noted that the lowest voltage (50) occurred between 7:30 and 7:45 a.m., and that the highest voltage was recorded about 1 hour earlier. Another low point (100 volts) was reached just before noon. The corresponding chart taken in the substation showed that a practically constant voltage of about 294 volts was maintained during the period in which the low readings were obtained. This indicates a drop of 244 volts, or 83 percent, between the substation and the inside meter station at the time that the minimum voltage was recorded. The chart clearly shows how cessation of activities for the lunch hour was reflected in the high voltage recorded from 12:10 to 12:25 p.m. A chart taken at the same place after the changes in the power system were completed showed that where formerly the voltage was under 250 during the entire 24 hours, except for momentary swings of the needle, the voltage now was between 250 and 300 or else off scale.

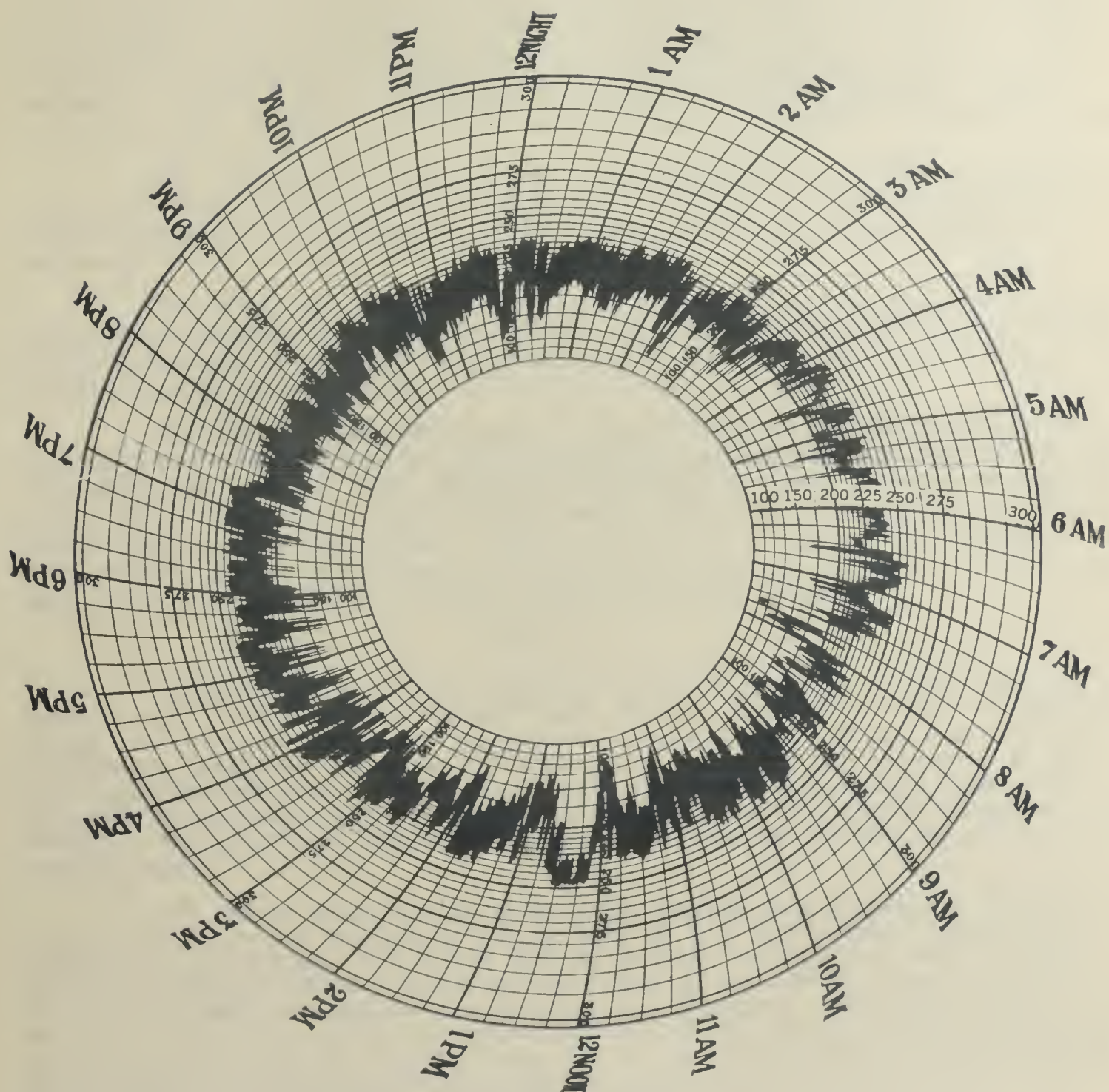


Figure 3.- Typical voltage chart, mine G.

This mine was operated by the same company that operated mine G, and changes in the power system at Mine H also were made about the same time as those made in the other mine. Specifically, these changes were as follows: (1) A substation with a synchronous rotary converter of 200-kw. capacity was added to the system near the active workings, and (2) the feeder from this substation was extended in the mine to connect with the existing feeder. Prior to these changes, the power supply consisted of a 300-kw. 275-volt synchronous rotary converter on the surface. Advantage was taken of the opportunity to secure voltmeter charts before the changes were made and again after they were completed.

Four places were selected for installation of the meters in securing voltage charts. One meter was connected to the switchboard in the 300-kw. substation and three were connected to the trolley system at various points in the mine.

The first set of charts showed that the voltage was maintained at a practically constant value between 295 and 298 volts. After the changes in the power system were completed, the voltage at the substation was less stable and varied from 285 to 300 volts.

The charts taken underground indicated that in general, voltage conditions were improved in remote sections of the mine by the addition of the new substation and feeder. There was one point, however, where voltages of 150, representing drops of 45 to 50 percent were recorded both before and after the changes were made in the system. This point was at the meter station most distant from the new substation, and being near the mine entrance had least need of voltage improvement since haulage only was affected.

Mine I

The last of the group of mines visited in securing voltage data was supplied with direct-current energy from a surface power plant near the hoisting shaft. In this plant were two synchronous converters of 300-kw. capacity and one 400-kw. generator driven by a steam engine. A meter was connected to the direct-current bus bars in the power plant, and three meters were taken into the mine for connection between rail and trolley at points remote from the generators. The points selected were within 50 to 100 feet of the end of the trolley wire.

The charts taken in the power plant showed that although a value of 275 volts or more was maintained during most of the test period, there was a period when the voltage fell below 250 and another time when it rose to 300 volts and remained at that value for more than an hour. The maximum variation at the switchboard, therefore was greater than 50 volts. The charts taken in the mine revealed sustained low voltage in all three places, the minimum being about 100 volts, or a loss of at least 150 volts. The records showed intervals of 15 minutes to half an hour during which the average voltage was under 150. It was below 200 volts for even greater intervals.

Discussion of Results

One authority states that the drop should not exceed 12-1/2 percent. On the other hand, a large coal operator has set 200 volts as being the minimum allowable for any location in his mines. Based on 295 volts maintained at his substations, this minimum figure would permit a drop of 32 percent.

In determining whether a drop of this amount is justifiable economically, each operator should decide for himself whether the cost of wasted electrical energy, the cost of repairs to damaged equipment, the cost of lowered production resulting from delays and slower moving equipment, and the cost of accidents can be offset by the cost of installing and maintaining adequate power circuits, sufficient generating capacity, and any other expense that might be incurred in securing "good voltage."

Small fluctuations in voltage have a marked effect upon the illumination given by lamps. Therefore, it is especially important in underground work to keep the drop at a low value to insure adequate illumination for safe operation of locomotives, for satisfactory work in shops, safety around moving machinery, and good visibility of lighted signals and track switches.

The trend toward "double shifting" of mining machinery offers still another reason for keeping voltage up. When machines were operated for only one shift in 24 hours, the time between shifts permitted cooling of armatures, field coils, and resistors. However, when another shift immediately follows the first one, little or no cooling can take place. Consequently, if the voltage has been low during the first shift it is not unlikely that safe operating temperatures have been exceeded, and so, without the advantage of cooling before the next shift begins, the machine may be seriously damaged.

From the data given for the nine mines investigated, it will be seen that the 12-1/2 percent figure previously cited for maximum drop was exceeded even in mine B, where the best voltage conditions were found. In this case, no doubt, the drop could have been made less than 12-1/2 percent by improving the return circuit through more efficient track bonding.

REMEDIES FOR LOW VOLTAGE

Many remedies, some temporary and some permanent, have been adopted in seeking relief from the difficulties that arise from low voltage. At certain mines, where hauling and cutting have been carried on during the same shift, it has been found advantageous to separate these two major loads and to do hauling on the day shift and cutting on the night shift to avoid demands beyond the capacity of the generating equipment. If pumping constitutes an appreciable proportion of the direct-current load, it is sometimes possible to change the hours of pumping to that period in the day when the load is lightest.

In the larger mines, where considerable heavy haulage is necessary, a system of dispatching is usually adopted. This may be intended primarily to permit trips to proceed safely and at regular intervals to the tippie or other point of discharge, but if dispatching is done properly, the movement of trips can also be regulated to prevent the heavy demands of the various main-line haulage locomotives from occurring simultaneously. This point may be illustrated from observations made at one of the mines where voltage charts were made.

Two of the loaded trips in this mine, drawn by 10-ton tandem locomotives, daily stopped at a point near the dispatcher's office while the crews gathered in his office to eat their noon lunch. Following this period, both trips were permitted to start together for the dumping point despite the fact that the grades were against them. The resulting effect on the voltage of the system was shown by the steep decline of the curves drawn by the recording instruments. If the dispatcher had maintained a 10- or 15- minute headway between these trips, it is probable that the voltage drop would have been reduced considerably. Control of the movement of trips by the use of automatic sectionalizing circuit breakers can be arranged to prevent occurrences such as the foregoing. With these breakers installed at suitable points, properly set as to maximum load to be carried and correctly adjusted to remain open for a predetermined interval once they have opened, it is possible to keep generating equipment from being overloaded by locomotives operated on improper schedules. One manufacturer of automatic equipment offers a system of dispatching in which the sectionalizing breakers can be operated by push buttons in the dispatcher's office. In this way, the movement of trips can be placed definitely under his control.

Although raising the voltage above normal at the generating station does not eliminate power losses due to inadequate carrying capacity of the direct-current distribution system, it is often resorted to in order to make it possible to operate machinery in remote parts of the mine. At one mine where power was purchased, the conversion equipment consisted of synchronous rotary converters having a direct-current output rated at 275 volts. The voltage at different points in the mine was quite low, and to offset this condition the taps on the transformer secondaries were changed to increase the voltage impressed on the converters. This change raised the direct-current voltage at the bus bars in the substations to 310 or more.

One possible objection to this procedure is that insulation that may have been weakened by overheating during periods of low voltage may break down when the higher voltage is applied to the equipment, especially that equipment operated near the substations. In addition, the higher voltage increases the shock hazard.

For many years the maximum size of trolley wire used was 4/0 A.W.G., but as larger capacity mine cars and larger haulage units were developed, a wire of greater carrying capacity was required. This led to the introduction of the 6/0 A.W.G. size, which is used extensively in mines having heavy haulage; however, even this size is being superseded by what is known as the "no. 9 section." The latter was practically double the cross-section area of the 4/0 wire.

Along with the stepping-up of trolley-wire size, it has been necessary to use larger rails to provide for faster and safer transportation with the heavier locomotives and cars. Where the larger rails, trolley wires, and feeders have been adopted, it has been possible to gain some improvement in voltage conditions. Further improvement has been found to be attainable through the welding of rail joints to increase conductivity over that ordinarily secured with the usual bonded joints.

As mine workings approach the property boundary lines, it may not be possible, even with larger rails, feeders, and trolley wire, to maintain satisfactory voltage conditions at remote points when feeder and trolley lines exceed 1-1/2 miles in length, unless an excessive amount of copper is installed or else substations are moved nearer the load centers. Instead of keeping several generators at one point, it is usually found that voltage conditions can be considerably improved by placing the generators in separate substations close to load centers.

Storage batteries floating across the line offer another method for raising the voltage at distant points in a mine. Locomotive batteries that were no longer satisfactory for haulage duty have been found suitable for the purpose of improving voltage when the life of the mine, high demand charge, and other conditions did not justify the purchase of additional generating equipment.

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UNITED STATES BUREAU OF MINES
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REPORT OF INVESTIGATIONS

COOPERATIVE FUEL RESEARCH MOTOR-GASOLINE SURVEY,
WINTER 1936-37



COMPILED BY

E. C. LANE

REPORT OF INVESTIGATIONS

DEPARTMENT OF THE INTERIOR - BUREAU OF MINES

COOPERATIVE FUEL RESEARCH MOTOR GASOLINE-SURVEY, WINTER 1936-37^{1/}

Compiled by E. C. Lane^{2/}

INTRODUCTION

This is the third in a series of reports on properties of commercial motor fuels, made in accordance with a cooperative agreement between the Cooperative Fuel Research (C. F. R.) Committee and the Bureau of Mines of the Department of the Interior. The first report gave data on motor gasolines sold during the winter of 1935-36.^{3/} The second report gave the same data for motor gasolines sold during the summer of 1936.^{4/}

As was the case with the first cooperative gasoline survey, reports were received for some areas from only one company and in other areas from two companies that did not in all instances report on the same brands of gasoline. In still other areas reports were received from five or six companies on the same brands of gasoline. Thus, it can be seen that data on gasolines are unequal in regard to the number of reports that were averaged in calculating the data shown in table 1. The data from the Pacific Coast were averaged by the Pacific Coast technical group before submission and are published exactly as they were submitted.

ACKNOWLEDGMENTS

The following companies contributed data for this gasoline survey: Continental Oil Co., Empire Oil and Refining Co., Gulf Refining Co., Humble Oil and Refining Co., Phillips Petroleum Co., Shell Petroleum Corporation, Sinclair Refining Co., Standard Oil Development Co. (N. J.), Standard Oil Co. of Ind., Tide Water Associated Oil Co., The Atlantic Refining Co., The Pure Oil Co., and the Pacific Coast Technical Group representing the major petroleum companies marketing in the Pacific Coast area. It is a pleasure

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- ^{1/} The Bureau of Mines will welcome reprinting of this paper, provided the following footnote acknowledgment is used: "Reprinted from U. S. Bureau of Mines Report of Investigations 3348."
^{2/} Assistant petroleum chemist, U. S. Bureau of Mines.
^{3/} Lane, E. C., and Kraemer, A. J., Cooperative Fuel Research Motor-Gasoline Survey, Winter 1935-36: Rept. of Investigations 3311, Bureau of Mines, September 1936, 63 pp.
^{4/} Lane, E. C., Cooperative Fuel Research Motor-Gasoline Survey, Summer 1936: Rept. of Investigations 3335, Bureau of Mines, January 1937, 61 pp.

to acknowledge the assistance of these companies and also the helpful co-operation of C. B. Veal, secretary of the C. F. R. Committee.

EXPLANATION OF TABLES

Table 1 gives data on gravity, sulphur content, Reid vapor pressure, copper-strip corrosion test, A. S. T. M. octane number, and distillation characteristics for commercial gasolines sold in 21 marketing areas in the United States during the winter of 1936-37. Item numbers are assigned to each brand in each area as a convenience in referring to the data. An item number in one part of the table has no relation to the same item number in another part of the table. Averages for each of the grades of gasoline are given at the foot of the sections of table 1, with data for each area.

The averages of table 1 are given in tables 2, 3, and 4, in which the data of the survey are summarized. In table 5 the data of the present survey are summarized in a statement of the minimum and maximum figure for each characteristic of each of the three groups of gasolines and in a range of figures that covers 90 percent of the items in each group. Table 6 is identical with table 5 of Report of Investigations 3335^{5/} and table 7 is identical with table 10 of Report of Investigations 3311^{6/}. These tables are included in this report for comparison with table 5.

DISCUSSION OF RESULTS

Octane Number

The octane ratings of the regular and premium grades of gasoline, as shown by a comparison of the data in table 5 and table 6, indicate somewhat higher values during the winter of 1936-37 than for the winter of 1935-36. The third-grade gasolines lie in substantially the same range for the two periods.

Volatility

The range of vapor pressures in which 90 percent of the samples of regular and premium-priced gasolines are included is within narrower limits than for the corresponding samples during the winter of 1935-36, notably in the higher figure for the minimum points. The minimum for the third-grade samples, however, reached a new low figure, while the maximum points are considerably lower for the present winter survey.

The range of temperatures in the distillation also is considerably narrower in table 5 of this survey than in table 10 of Report of Investigations 3311. The end points of 90 percent of the gasolines in the regular and premium-price gasolines have a slightly greater range than those of the previous winter. The minimum points have been raised and the maximum points lowered for the regular and premium-priced gasolines in comparison with those for the winter of 1935-36.

^{5/} See footnote 4.

^{6/} See footnote 3.

SUMMARY

This survey of motor gasoline sold to the general public through service stations in marketing areas throughout the United States during the winter of 1936-37 is the third that has been published pursuant to a co-operative agreement between the United States Bureau of Mines and the Co-operative Fuel Research Committee.

Characteristics of regular-price, premium-price, and third-grade motor gasolines are reported, based upon inspections made in oil-company laboratories and submitted to the Bureau of Mines for compilation and comment.

This survey indicates that the range of variations with respect to octane rating and volatility in each of the major price groups is becoming smaller. This tendency is a manifestation of the progressive improvement in gasoline uniformity that has been noted in previous surveys of the Bureau of Mines.

TABLE 1. - Motor-gasoline survey, winter 1936-37
(Data for samples from different cities)

BOSTON, MASS.

Regular-price gasoline

Item	Gravity		Sulphur, %	Reid vapor pressure, lb.	Copper- strip test	ASTM octane number
	Specific	API				
1	0.726	63.5	0.094	11.4	Neg.	70.5
2	.727	63.1	.044	10.9	do.	69.5
3	.731	62.0	.035	9.3	do.	70.8
4	.733	61.5	.085	11.2	do.	70.5
5	.734	61.3	.068	9.9	do.	70.0
6	.734	61.2	.043	8.9	do.	69.5
7	.735	61.0	.064	10.8	do.	68.8
8	.736	60.9	.125	10.3	do.	69.5
9	.737	60.6	.076	9.6	do.	69.8
10	.738	60.3	.064	9.5	do.	70.0
11	.742	59.3	.043	9.7	do.	69.0
12	.783	49.2	.11	7.4	do.	74.0
Average	.738	60.3	.071	9.9	do.	70.2

Premium-price gasoline

1	0.717	65.8	0.088	10.3	-	80.5
2	.725	63.6	.03	11.4	Neg.	77.0
3	.732	61.9	.035	9.0	-	75.8
4	.733	61.6	.042	8.8	Neg.	78.5
5	.733	61.5	.053	9.4	do.	78.5
6	.735	61.1	.051	9.5	do.	77.5
7	.735	61.1	.089	10.5	-	76.3
8	.736	60.9	.041	8.9	Neg.	78.5
9	.736	60.9	.063	8.4	-	78.3
10	.738	60.3	.086	9.2	-	75.5
11	.775	51.1	.131	8.1	Neg.	75.5
Average	.736	60.9	.064	9.4	do.	77.7

Third-grade gasoline

1	0.729	62.5	-	10.7	Neg.	64.0
2	.729	62.5	-	9.7	-	59.0
3	.732	61.8	-	10.5	Neg.	63.0
4	.733	61.4	-	10.2	do.	64.0
5	.734	61.3	-	7.7	do.	61.0
6	.737	60.4	-	9.2	do.	64.0
7	.738	60.1	-	7.2	do.	64.0
8	.749	57.4	-	7.8	do.	65.0
Average	.738	60.1	-	9.1	do.	63.0

TABLE 1. - Motor-gasoline survey, winter 1936-37 - Continued
(Data for samples from different cities)

BOSTON, MASS.

Regular-price gasoline (continued)

I.B.P.	Distillation range, °F.							Percent		Item
	Percent recovered						End point	Resid.	Loss	
	10	20	30	50	70	90				
85	129	157	183	240	278	344	414	1.1	3.2	1
86	126	153	180	238	290	356	412	1.1	1.9	2
93	147	174	200	243	282	342	398	1.1	1.7	3
86	124	149	175	237	289	355	397	1.1	1.7	4
91	134	163	191	241	282	344	392	1.1	1.9	5
93	143	169	197	241	284	347	395	1.1	1.4	6
91	134	163	192	242	285	343	385	1.1	2.2	7
91	133	162	191	238	283	350	396	1.1	2.2	8
95	139	168	196	244	287	347	395	1.1	1.7	9
94	137	169	199	252	301	357	399	1.1	1.9	10
88	138	169	200	247	294	361	401	1.1	2.4	11
107	155	167	187	199	245	336	394	1.1	.9	12
92	137	164	191	239	283	349	398	1.1	1.9	Average

Premium-price gasoline (continued)

93	140	159	176	203	228	279	361	1.2	1.6	1
90	135	157	180	218	263	341	415	1.2	2.3	2
96	142	168	190	226	263	315	364	1.0	1.5	3
97	147	175	201	244	284	344	396	1.2	1.9	4
97	143	171	197	241	286	345	395	1.1	1.4	5
95	145	173	197	243	284	347	397	1.2	1.3	6
90	128	156	180	237	291	357	400	1.1	1.7	7
97	144	170	193	243	286	345	389	1.1	1.5	8
94	145	168	194	233	270	327	387	1.1	.7	9
95	141	168	193	233	275	336	394	1.1	1.2	10
99	147	165	179	200	240	323	385	1.1	.7	11
95	142	166	190	229	270	333	390	1.1	1.4	Average

Third-grade gasoline (continued)

95	133	160	182	226	273	351	417	1.0	2.5	1
90	140	171	197	239	289	351	408	1.0	3.0	2
86	120	149	179	239	289	347	374	1.0	3.5	3
90	129	166	194	243	281	347	401	1.0	2.5	4
90	151	183	208	252	294	356	392	1.0	2.5	5
93	143	173	193	233	277	331	392	1.0	2.0	6
97	150	173	204	245	284	338	390	1.0	1.0	7
93	147	181	212	264	309	369	421	1.0	2.0	8
92	140	170	197	243	287	349	399	1.0	2.4	Average

TABLE 1. - Motor gasoline survey, winter 1936-37 - Continued
(Data for samples from different cities)

NEW YORK, N. Y.

Regular-price gasoline

Item	Gravity		Sulphur, %	Reid vapor pressure, lb.	Copper- Strip test	ASTM octane number
	Specific	°API				
1	0.725	63.8	0.039	11.2	Neg.	69.8
2	.727	63.1	.047	10.5	do.	70.0
3	.730	62.4	.039	10.9	do.	70.3
4	.731	62.2	.051	11.2	do.	73.0
5	.732	61.9	.041	9.9	-	71.0
6	.732	61.9	.066	9.5	Neg.	70.0
7	.733	61.5	.081	10.0	do.	71.0
8	.735	61.1	.074	10.9	do.	71.0
9	.735	61.0	.038	9.8	-	71.5
10	.736	60.9	.069	10.9	Neg.	68.5
11	.736	60.7	.069	10.7	do.	70.5
12	.737	60.6	.089	9.5	do.	70.5
13	.738	60.3	.071	9.9	do.	70.5
14	.739	60.0	.124	9.6	do.	70.3
15	.740	59.7	.045	11.3	do.	70.5
Average	.734	61.4	.063	10.4	do.	70.6

Premium-price gasoline

1	0.721	64.7	0.030	10.9	Neg.	78.0
2	.724	64.0	.038	9.2	do.	78.3
3	.724	63.9	.06	9.0	do.	80.0
4	.727	63.1	.042	9.5	do.	76.0
5	.728	63.0	.047	10.1	do.	78.3
6	.728	62.8	.047	9.4	do.	77.5
7	.730	62.2	.075	9.4	do.	77.3
8	.731	62.0	.057	9.0	do.	83.0
9	.738	60.3	.068	9.3	do.	77.5
10	.738	60.3	.084	10.3	-	76.0
11	.738	60.2	.067	9.0	-	77.0
12	.739	60.1	.042	9.1	Neg.	77.0
13	.783	49.3	.144	7.3	do.	76.0
Average	.734	61.2	.062	9.3	do.	77.8

TABLE 1. - Motor-gasoline survey, winter 1936-37 - Continued
(Data for samples from different cities)

NEW YORK, N. Y.

Regular-price gasoline (continued)

I.B.F.	Distillation range, °F.									Item
	Percent recovered						End point	Percent		
	10	20	30	50	70	90		Resid.	Loss	
87	129	158	184	236	282	343	391	1.2	1.9	1
90	132	161	189	239	288	349	396	1.1	1.7	2
91	137	168	196	247	297	360	411	1.2	2.6	3
86	135	165	192	235	277	336	384	1.3	1.6	4
92	136	164	194	249	295	352	390	1.1	2.4	5
92	139	166	195	243	293	352	396	1.0	1.5	6
89	130	159	187	237	283	346	413	1.3	1.9	7
85	123	150	179	238	293	355	397	1.0	2.1	8
93	136	167	196	243	286	350	399	1.2	1.6	9
93	136	167	196	244	286	341	387	1.0	2.0	10
83	130	159	191	243	284	340	382	1.0	2.0	11
90	131	168	199	249	295	354	397	1.2	1.5	12
91	138	167	196	244	287	343	396	1.1	1.4	13
91	137	166	193	238	284	348	399	1.2	1.9	14
86	131	165	196	244	289	359	399	1.1	3.0	15
89	133	163	192	255	288	349	396	1.1	1.9	Average

Premium-price gasoline (continued)

95	136	157	176	217	255	334	410	1.2	1.9	1
93	144	168	188	221	251	296	362	1.1	1.7	2
91	133	154	173	207	243	301	383	1.1	1.3	3
101	138	165	192	239	287	343	383	1.2	1.8	4
91	133	160	188	238	279	339	385	1.1	1.7	5
95	136	163	190	235	280	341	388	1.2	1.3	6
90	140	168	197	240	279	332	389	1.1	1.9	7
97	140	167	192	239	284	342	388	1.2	1.8	8
95	142	187	196	239	280	343	390	1.4	1.1	9
89	129	156	185	241	293	354	396	1.0	1.9	10
96	140	169	196	244	288	347	391	1.0	2.0	11
95	143	173	199	251	299	356	410	1.3	1.2	12
100	150	168	181	201	243	333	383	1.0	4.0	13
94	139	166	189	232	274	336	389	1.2	1.8	Average

TABLE 1. - Motor-gasoline survey, winter 1936-37 - Continued
(Data for samples from different cities)

PHILADELPHIA, PA.

Regular-price gasoline

Item	Gravity		Sulphur, %	Reid vapor pressure, lb.	Copper- strip test	ASTM octane number
	Specific	°API				
1	0.724	64.0	0.035	10.0	Neg.	70.7
2	.728	63.0	.030	10.1	do.	71.9
3	.731	62.1	.034	10.0	do.	70.5
4	.731	62.0	.026	9.7	do.	71.5
5	.732	61.7	.030	9.0	do.	70.6
6	.734	61.2	.055	10.6	do.	71.4
7	.736	60.7	.166	8.8	do.	69.7
8	.742	59.3	.044	11.0	do.	70.4
9	.742	59.2	.044	10.3	do.	70.7
10	.742	59.2	.042	10.2	do.	70.7
Average	.734	61.2	.051	10.0	do.	70.8

Premium-price gasoline

1	0.722	64.5	0.021	10.4	Neg.	77.4
2	.724	64.0	.033	10.0	do.	78.4
3	.725	63.8	.042	8.8	do.	78.0
4	.725	63.6	.049	9.0	do.	78.2
5	.731	62.0	.043	8.8	do.	76.9
6	.735	61.1	.039	9.3	do.	76.8
7	.739	60.1	.040	8.4	do.	77.4
8	.739	59.9	.022	8.2	do.	78.7
9	.743	59.0	.043	10.0	do.	77.4
10	.743	59.0	.043	9.5	do.	77.0
Average	.732	61.7	.034	9.2	do.	77.6

Third-grade gasoline

1	0.724	63.8	0.024	8.2	Neg.	58.2
2	.731	62.1	.033	9.4	do.	65.0
3	.743	58.9	.041	8.6	do.	62.5
4	.744	58.8	.070	7.8	do.	60.9
Average	.736	60.9	.042	8.5	do.	61.7

TABLE 1. - Motor-gasoline survey, winter 1936-37 - Continued
(Data for samples from different cities)

PHILADELPHIA, PA.

Regular-price gasoline (continued)

I.B.P.	Distillation range, °F.							Percent		Item
	Percent recovered						End point			
	10	20	30	50	70	90		Resid.	Loss	
93	128	155	184	231	269	326	378	1.1	1.6	1
90	127	156	186	237	287	354	420	1.1	2.1	2
90	135	165	195	245	285	336	378	1.2	1.9	3
93	132	163	193	246	291	353	396	3.0	1.5	4
91	132	163	195	249	294	348	404	1.1	1.3	5
83	130	161	194	241	280	342	379	1.0	2.0	6
93	134	159	182	223	264	328	402	.9	1.2	7
87	135	169	201	249	294	360	399	1.2	2.6	8
86	128	161	195	249	294	348	391	1.2	1.5	9
88	131	163	194	243	289	340	390	1.0	1.0	10
89	131	161	192	241	285	344	394	1.3	1.7	Average

Premium-price gasoline (continued)

87	129	153	174	213	255	340	416	1.0	2.0	1
88	130	158	186	232	274	335	378	1.0	1.5	2
96	135	158	180	217	253	311	366	1.0	1.0	3
90	132	152	171	201	230	286	366	1.0	1.0	4
85	137	170	197	237	266	320	381	1.0	1.0	5
85	131	163	194	247	297	349	408	1.0	1.0	6
90	136	166	194	241	282	349	401	1.0	1.0	7
93	135	162	190	237	286	350	402	1.0	1.0	8
80	124	156	186	229	274	332	373	1.0	2.0	9
82	125	155	188	244	292	343	389	1.0	1.0	10
88	131	159	186	230	271	332	388	1.0	1.3	Average

Third-grade gasoline (continued)

96	140	166	193	235	278	330	371	1.0	1.5	1
93	134	156	178	218	263	342	419	1.0	1.0	2
88	137	170	200	247	294	362	408	1.0	1.5	3
88	138	172	205	256	302	350	392	1.0	1.5	4
91	137	166	194	239	284	346	398	1.0	1.4	Average

TABLE 1. - Motor-gasoline survey, winter 1936-37 - Continued
(Data for samples from different cities)

BALTIMORE, MD.

Regular-price gasoline

Item	Gravity		Sulphur, %	Reid vapor pressure, lb.	Copper- strip test	ASTM octane number
	Specific	°API				
1	0.724	63.9	0.033	10.0	Neg.	68.6
2	.727	63.1	.079	11.0	do.	71.0
3	.727	63.0	.038	10.2	do.	69.7
4	.729	62.6	.007	10.3	do.	70.5
5	.732	61.9	.062	9.1	do.	70.0
6	.733	61.6	.109	10.0	do.	69.8
7	.735	61.0	.038	10.4	do.	72.0
8	.736	60.9	.075	10.5	do.	71.0
9	.736	60.8	.042	10.5	do.	70.8
10	.737	60.4	.048	10.5	do.	70.3
11	.740	59.8	.037	10.6	do.	70.3
Average	.732	61.7	.052	10.3	do.	70.4

Premium-price gasoline

1	0.713	67.1	0.088	12.2	Neg.	80.5
2	.723	64.1	.031	9.2	do.	77.5
3	.733	61.6	.043	8.8	do.	76.0
4	.737	60.6	.086	8.3	do.	77.0
5	.738	60.2	.055	10.3	do.	76.5
6	.766	53.2	.160	7.4	do.	75.0
7	.776	50.8	.165	9.5	do.	77.5
8	.777	50.5	.159	7.7	do.	76.0
Average	.745	58.5	.098	9.2	do.	77.0

TABLE 1. - Motor-gasoline survey, winter 1936-37 - Continued
(Data for samples from different cities)

BALTIMORE, MD.

Regular-price gasoline (continued)

I.B.P.	Distillation range, °F.							Percent		Item
	Percent recovered						End point	Resid.	Loss	
	10	20	30	50	70	90				
86	122	150	179	230	275	343	392	1.0	2.0	1
86	125	153	179	229	278	339	391	1.1	2.6	2
88	129	157	181	228	275	338	391	1.0	1.2	3
88	125	155	182	234	280	350	396	1.0	2.0	4
93	138	168	197	244	292	358	422	1.1	1.7	5
91	133	163	187	232	274	339	397	1.1	2.5	6
92	130	163	192	245	290	361	400	1.0	3.0	7
86	131	163	195	244	283	346	390	1.1	2.0	8
87	126	157	186	243	290	352	396	1.1	1.9	9
86	127	158	190	244	283	343	397	1.1	2.6	10
87	136	167	198	246	293	357	400	1.1	2.9	11
88	129	159	188	238	284	348	397	1.1	2.2	Average

Premium-price gasoline (continued)

89	127	149	169	201	232	277	335	1.4	2.6	1
95	140	161	179	217	269	338	413	1.4	1.1	2
99	151	178	203	248	295	353	394	1.1	.9	3
101	147	174	199	241	282	345	405	1.2	.8	4
88	129	160	189	245	289	347	390	1.1	.9	5
106	151	167	179	203	246	329	389	.6	.9	6
99	145	165	178	201	245	333	387	1.2	1.3	7
104	149	167	178	196	230	325	392	1.1	.4	8
98	142	165	184	219	261	331	388	1.1	1.1	Average

TABLE 1. - Motor-gasoline survey, winter 1936-37 - Continued
(Data for samples from different cities)

PITTSBURGH, PA.

Regular-price gasoline

Item	Gravity		Sulphur, %	Reid vapor pressure, lb.	Copper- strip test	ASTM octane number
	Specific	°API				
1	0.720	65.0	0.070	8.0	Neg.	70.5
2	.723	64.1	.051	10.8	do.	71.0
3	.725	63.8	.037	10.7	do.	69.8
4	.730	62.4	.100	-	do.	71.0
5	.733	61.6	.070	9.0	do.	70.3
6	.733	61.6	.080	-	Pos.	70.0
7	.734	61.4	.059	10.2	Neg.	69.8
8	.735	61.1	.087	9.0	do.	71.5
9	.735	61.0	.072	9.9	do.	72.3
10	.736	60.9	.063	11.5	do.	71.0
11	.737	60.6	.030	-	do.	70.0
12	.737	60.4	.078	-	do.	70.5
13	.741	59.6	.066	10.8	do.	70.2
14	.741	59.4	.047	9.8	do.	70.0
Average	.733	61.6	.065	10.0	do.	70.6

Premium-price gasoline

1	0.719	65.4	0.060	10.5	Neg.	79.0
2	.719	65.3	.082	8.9	do.	79.0
3	.722	64.4	.056	10.2	do.	79.0
4	.728	62.9	.066	11.5	do.	79.0
5	.729	62.6	.054	8.6	do.	76.0
6	.739	60.0	.044	8.5	do.	78.0
7	.744	58.7	.061	9.1	do.	77.5
8	.744	58.7	.095	7.6	do.	76.0
9	.776	50.8	.105	8.4	do.	77.5
Average	.735	61.0	.069	9.3	do.	77.9

TABLE 1. - Motor-gasoline survey, winter 1936-37 - Continued
(Data for samples from different cities)

PITTSBURGH, PA.

Regular-price gasoline (continued)

I.B.P.	Distillation range, °F.									Item
	Percent recovered						End point	Percent		
	10	20	30	50	70	90		Resid.	Loss	
102	160	185	203	232	259	305	359	1.0	1.5	1
93	140	171	197	241	282	341	381	1.3	1.7	2
92	130	158	180	224	265	322	373	1.0	1.8	3
86	127	154	181	233	277	338	398	1.0	1.0	4
87	136	164	191	235	282	347	399	1.0	1.1	5
83	124	145	164	193	225	287	342	1.0	1.0	6
90	136	166	193	241	287	351	405	1.1	1.9	7
91	137	165	192	241	287	345	394	1.2	.8	8
87	130	155	181	224	297	326	374	1.1	1.4	9
88	137	162	193	243	283	345	391	1.1	1.5	10
95	146	175	200	247	298	357	396	1.0	1.0	11
70	132	162	190	240	285	345	396	1.0	2.0	12
89	133	166	196	245	283	354	402	1.1	1.6	13
95	142	171	197	243	287	354	403	1.4	1.6	14
89	136	164	190	234	279	344	387	1.0	1.4	Average

Premium-price gasoline (continued)

89	127	151	173	218	266	340	407	1.2	1.3	1
101	140	158	174	209	243	297	367	1.4	1.1	2
93	138	160	181	217	253	323	379	1.2	.8	3
88	131	158	187	232	271	329	377	1.3	1.7	4
95	143	169	192	237	279	388	385	1.1	.9	5
95	147	176	199	239	279	343	393	1.1	.9	6
91	138	169	197	250	293	347	394	1.1	.9	7
101	155	185	209	251	297	361	419	1.3	.7	8
106	151	169	181	201	232	305	369	1.0	.5	9
95	141	166	188	228	268	337	388	1.2	1.0	Average

(Data for samples from different cities)

CLEVELAND, OHIO

Regular-price gasoline

Item	Gravity		Sulphur, %	Reid vapor pressure, lb.	Copper- strip test	ASTM octane number
	Specific	API				
1	0.716	66.2	0.050	9.5	Neg.	70.4
2	.720	65.0	.040	-	do.	71.0
3	.726	63.4	.035	10.9	do.	70.2
4	.727	63.2	.060	-	do.	71.0
5	.727	63.0	.038	9.4	do.	70.1
6	.730	62.4	.080	-	do.	70.0
7	.730	62.3	.056	9.4	do.	70.1
8	.730	62.2	.050	-	do.	70.0
9	.731	62.0	.025	8.3	do.	70.0
10	.732	61.9	.070	11.6	do.	70.5
11	.732	61.8	.035	9.0	do.	69.7
12	.732	61.7	.060	-	do.	69.0
13	.737	60.6	.075	9.3	do.	69.9
14	.737	60.5	.050	-	do.	70.0
15	.737	60.5	.131	10.6	do.	69.3
16	.743	59.0	.052	10.8	-	70.0
Average	.731	62.2	.057	9.9	Neg.	70.1

Premium-price gasoline

1	0.713	66.9	0.030	9.9	Neg.	76.0
2	.714	66.7	.030	-	do.	76.0
3	.714	66.7	.030	-	do.	76.0
4	.715	66.5	.030	-	do.	76.0
5	.716	66.0	.030	-	do.	75.0
6	.718	65.6	.028	9.5	do.	78.0
7	.719	65.4	.017	9.8	do.	77.0
8	.720	65.0	.040	-	do.	74.0
9	.727	63.2	.050	-	do.	78.0
10	.727	63.0	.050	10.5	do.	78.0
11	.729	62.7	.040	-	do.	76.0
12	.733	61.5	.055	11.0	do.	79.0
13	.734	61.4	.084	9.5	do.	78.0
14	.773	51.7	.190	-	do.	76.0
15	.775	51.2	.110	-	do.	75.0
Average	.728	62.9	.054	10.0	do.	76.5

Third-grade gasoline

1	0.721	64.8	0.070	-	Neg.	50.0
2	.722	64.4	.050	-	do.	62.0
3	.727	63.1	.050	-	do.	50.0
4	.728	62.8	.030	-	do.	66.0
5	.729	62.6	.030	-	do.	52.0
6	.730	62.3	.030	-	do.	52.0
7	.732	61.7	.016	8.4	do.	53.0
8	.732	61.7	.040	5.1	do.	51.0
9	.734	61.3	.037	7.4	do.	51.7
10	.737	60.6	.040	-	do.	54.0
11	.740	59.8	.026	5.3	do.	52.0
12	.741	59.4	.040	-	do.	55.0
Average	.731	62.0	.038	6.6	do.	54.1

TABLE 1. - Motor-gasoline survey, winter 1936-37 - Continued

R.I.3348

(Data for samples from different cities)

CLEVELAND, OHIO

Regular-price gasoline (continued)

Regular price gasoline (continued)										
I.B.P.	Distillation range, °F.							Percent		Item
	Percent recovered						End point	Resid.	Loss	
	10	20	30	50	70	90				
90	129	154	184	238	292	355	394	1.1	1.2	1
87	130	158	186	230	276	342	400	1.0	1.0	2
87	127	155	181	232	274	363	430	1.2	2.6	3
86	127	156	182	226	274	362	443	1.0	1.0	4
92	134	164	194	241	284	339	389	1.2	2.3	5
87	126	150	173	212	259	335	406	1.0	1.0	6
92	133	161	189	240	285	345	393	1.0	2.5	7
89	133	165	196	244	290	362	430	2.0	1.0	8
89	130	160	187	241	295	368	406	.9	2.6	9
87	132	163	193	245	288	346	384	1.0	2.1	10
92	135	165	194	246	292	353	398	1.1	1.4	11
95	143	170	196	242	284	341	397	1.0	1.0	12
91	137	169	198	245	292	354	398	1.0	2.8	13
85	124	156	186	238	283	340	390	1.0	1.0	14
88	121	153	183	233	287	357	396	1.1	2.7	15
92	137	170	201	249	289	360	404	1.0	2.3	16
89	123	161	189	237	284	351	404	1.1	1.8	Average

Premium-price gasoline (continued)

90	132	156	179	213	244	302	384	1.1	2.3	1
86	125	154	180	220	254	316	385	1.0	1.0	2
86	128	157	182	223	260	324	388	2.0	1.0	3
91	131	158	183	220	254	314	398	1.0	1.0	4
90	130	157	182	220	253	313	382	1.0	1.0	5
90	120	143	170	220	273	338	380	1.0	3.0	6
88	121	140	167	217	263	336	380	.9	3.1	7
98	140	165	186	220	254	313	398	1.0	1.0	8
88	133	159	183	223	262	316	376	1.0	1.0	9
91	132	160	187	233	272	342	386	1.0	2.0	10
87	126	152	180	235	288	350	390	1.0	1.0	11
86	120	146	176	230	282	350	386	1.1	2.9	12
91	132	160	190	237	285	352	392	.7	2.6	13
98	138	156	169	184	203	297	398	1.0	1.0	14
95	148	168	182	204	243	323	381	1.0	1.0	15
90	130	155	180	220	259	326	387	1.1	1.7	Average

Third-grade gasoline (continued)

98	161	188	209	239	268	314	368	1.0	1.0	1
97	146	167	188	223	258	317	394	1.0	1.0	2
102	152	184	210	251	295	352	406	1.0	1.0	3
92	133	162	189	236	279	339	391	1.0	1.0	4
104	156	184	206	240	276	337	427	1.0	1.0	5
98	143	177	207	248	284	344	437	1.0	1.0	6
88	142	180	212	259	307	365	404	1.1	2.9	7
102	165	195	222	262	299	347	409	1.5	1.5	8
100	149	182	209	249	286	353	426	1.0	1.5	9
104	165	190	210	245	280	345	429	1.0	1.0	10
105	161	190	215	259	305	368	408	1.6	2.6	11
99	150	182	214	266	315	372	420	1.0	1.0	12
99	152	182	208	248	288	346	410	1.1	1.4	Average

TABLE 1. - Motor-gasoline survey, winter 1936-37 - Continued
(Data for samples from different cities)

DETROIT, MICH.

Regular-price gasoline

Item	Gravity		Sulphur, %	Reid vapor pressure, lb.	Copper- strip test	ASTM octane number
	Specific	°API				
1	0.718	65.6	0.060	11.0	-	70.0
2	.722	64.5	.040	-	Neg.	70.0
3	.722	64.4	.060	10.9	do.	69.8
4	.723	64.3	.054	11.1	do.	68.5
5	.724	64.0	.040	10.5	do.	70.0
6	.727	63.1	.050	10.5	do.	70.0
7	.729	62.7	.047	10.5	do.	70.0
8	.729	62.6	.061	9.6	do.	69.8
9	.729	62.6	.063	10.3	do.	69.6
10	.732	61.7	.140	12.3	do.	69.9
Average	.726	63.5	.062	10.7	do.	69.8

Premium-price gasoline

1	0.711	67.5	-	11.4	-	77.5
2	.723	64.1	0.050	10.3	Neg.	75.3
3	.725	63.7	.050	10.1	do.	75.5
4	.729	62.7	.040	9.6	do.	76.3
5	.729	62.6	.040	11.0	do.	77.3
6	.730	62.4	.050	-	do.	74.0
7	.731	62.1	-	10.7	do.	77.5
8	.734	61.4	.052	9.3	do.	76.5
9	.736	60.8	.062	9.2	do.	75.3
Average	.727	63.0	.049	9.1	do.	76.1

Third-grade gasoline

1	0.730	62.2	-	7.8	-	52.0
2	.734	61.4	-	7.1	-	51.5
3	.735	61.0	-	4.7	Neg.	54.0
4	.737	60.6	-	8.1	do.	54.8
Average	.734	61.3	-	6.9	do.	53.1

TABLE 1. - Motor-gasoline survey, winter 1936-37 - Continued
(Data for samples from different cities)

DETROIT, MICH.

Regular-price gasoline (continued)										
I.B.P.	Distillation range, °F.							Percent		Item
	Percent recovered						End point	Resid.	Loss	
	10	20	30	50	70	90				
83	112	131	154	212	269	338	395	1.0	1.5	1
89	128	153	182	234	288	350	392	1.0	1.0	2
86	121	145	172	232	288	361	408	1.1	1.6	3
87	122	146	171	213	252	318	386	.9	2.0	4
88	125	149	176	238	292	363	410	1.1	1.6	5
88	124	150	180	234	286	348	390	1.1	1.9	6
88	130	156	183	238	291	366	407	1.0	3.0	7
91	133	162	187	232	277	360	396	1.0	1.7	8
88	131	160	186	233	279	341	384	1.1	2.1	9
84	123	150	177	226	278	354	390	1.1	3.4	10
87	125	150	177	229	280	350	396	1.0	2.0	Average

Premium-price gasoline (continued)

88	110	127	146	189	240	302	382	1.0	0.0	1
86	123	149	177	235	292	361	406	1.0	1.7	2
88	125	153	178	237	292	357	406	1.0	1.5	3
88	126	152	182	236	290	351	390	1.0	1.0	4
86	122	145	168	214	269	344	389	1.0	2.0	5
86	125	151	173	238	297	367	404	1.0	1.0	6
85	124	153	181	233	283	351	390	1.0	2.8	7
89	133	154	181	233	281	347	391	1.0	2.0	8
87	134	163	190	238	283	339	383	1.0	1.7	9
87	125	150	176	228	281	347	393	1.0	1.5	Average

Third-grade gasoline (continued)

92	148	178	204	250	298	350	398	1.0	1.0	1
96	148	180	207	253	302	352	404	1.0	0.0	2
116	172	192	211	244	280	341	398	1.0	1.0	3
94	150	184	214	253	302	356	421	1.0	1.5	4
100	155	184	209	251	296	350	405	1.0	.9	Average

TABLE 1. - Motor-gasoline survey, winter 1936-37 - Continued
(Data for samples from different cities)

ATLANTA, GA.

Regular-price gasoline

Item	Gravity		Sulphur, %	Reid vapor pressure, lb.	Copper- strip test	ASTM octane number
	Specific	°API				
1	0.729	62.6	0.015	9.6	Neg.	71.0
2	.729	62.6	.035	8.9	do.	70.0
3	.729	62.6	.050	-	do.	69.0
4	.729	62.6	.100	8.6	do.	70.6
5	.729	62.5	.029	8.4	do.	69.0
6	.730	62.4	.046	8.6	do.	69.4
7	.730	62.4	.050	-	do.	69.0
8	.736	60.9	.110	-	do.	68.0
9	.737	60.6	.052	8.5	do.	65.0
10	.737	60.5	.055	9.4	do.	68.0
11	.737	60.4	.059	8.9	do.	70.1
12	.742	59.3	.061	7.3	do.	69.0
13	.743	58.9	.110	-	do.	69.0
14	.744	58.7	.100	9.4	do.	69.3
15	.749	57.4	.036	8.4	do.	70.3
16	.753	56.4	.100	-	do.	68.0
Average	.736	60.7	.060	8.7	do.	69.0

Premium-price gasoline

1	0.718	65.7	0.007	7.3	Neg.	79.0
2	.729	62.5	.028	7.5	do.	75.0
3	.739	60.0	.039	8.5	do.	78.0
4	.745	58.4	.039	8.9	do.	77.0
Average	.732	61.7	.028	8.1	do.	77.3

Third-grade gasoline

1	0.726	63.4	0.030	9.0	Neg.	58.0
2	.729	62.7	.070	-	do.	65.0
3	.732	61.7	.030	-	do.	57.0
4	.733	61.6	.040	-	do.	58.0
5	.735	61.1	.055	9.0	do.	62.0
6	.736	60.8	.056	9.2	do.	69.0
7	.736	60.8	.059	9.3	do.	66.0
8	.740	59.7	.038	9.5	Pos.	67.0
9	.742	59.2	.050	-	Neg.	58.0
10	.744	58.7	.071	9.2	do.	68.0
11	.745	58.3	.098	8.4	do.	69.0
12	.746	58.2	.027	6.9	do.	65.0
13	.748	57.7	.040	-	do.	56.0
Average	.738	60.3	.051	8.8	do.	62.9

TABLE 1. - Motor-gasoline survey, winter 1936-37 - Continued
(Data for samples from different cities)

ATLANTA, GA.

Regular-price gasoline (continued)

I.B.P.	Distillation range, °F.						End point	Percent		Item
	Percent recovered							Resid.	Loss	
	10	20	30	50	70	90				
94	135	163	189	237	282	335	383	1.0	1.8	1
90	134	163	191	239	280	338	385	1.1	1.3	2
90	136	165	192	240	294	354	385	1.0	1.0	3
93	135	165	193	239	281	341	398	1.2	1.4	4
88	132	165	192	241	285	351	397	1.0	2.0	5
98	143	175	202	245	284	342	407	1.3	1.4	6
90	140	168	193	240	285	338	388	1.0	1.0	7
88	130	157	185	238	285	342	398	1.0	1.0	8
98	144	172	198	238	282	340	390	1.0	1.0	9
89	135	168	196	242	281	340	384	1.0	2.0	10
92	139	170	200	246	289	345	397	1.2	1.4	11
97	141	170	200	249	293	343	390	1.0	1.0	12
85	135	166	196	257	318	368	410	1.0	1.0	13
94	141	168	193	229	278	345	400	1.1	1.9	14
91	137	171	203	252	296	352	393	1.0	1.2	15
96	149	181	210	254	293	344	387	1.0	1.0	16
92	138	168	196	243	288	344	393	1.1	1.3	Average

Premium-price gasoline (continued)

99	142	160	178	206	237	287	378	1.0	1.0	1
92	135	165	191	244	288	340	392	1.0	1.5	2
92	135	167	194	242	281	336	396	1.0	1.0	3
84	127	162	194	250	295	352	394	1.0	2.0	4
92	135	164	189	235	275	329	390	1.0	1.4	Average

Third-grade gasoline (continued)

91	141	172	198	245	289	341	387	1.0	1.3	1
98	135	159	184	231	283	345	394	1.0	1.0	2
105	154	181	209	250	293	348	389	1.0	1.0	3
98	143	182	208	251	295	347	387	1.0	1.0	4
97	145	177	207	254	299	359	404	1.0	2.0	5
97	143	174	205	247	288	352	380	1.0	4.0	6
94	138	168	198	246	288	342	392	1.0	1.0	7
85	140	175	207	246	281	343	389	1.0	2.0	8
100	158	186	211	255	296	350	406	1.0	1.0	9
87	138	170	202	260	315	365	406	1.0	2.0	10
100	144	180	210	258	300	364	407	1.0	2.0	11
96	146	180	209	256	292	345	399	1.0	1.0	12
112	180	210	234	274	314	374	435	1.0	1.0	13
97	147	178	206	252	295	352	398	1.0	1.6	Average

R.I. 3348 TABLE 1. - Motor-gasoline survey, winter 1936-37 - Continued
(Data for samples from different cities)

CHICAGO, ILL.

Regular-price gasoline

Item	Gravity		Sulphur, %	Reid vapor pressure, lb.	Copper- strip test	ASTM octane number
	Specific	°API				
1	0.710	67.8	0.043	11.6	Neg.	70.1
2	.716	66.2	.049	12.0	do.	69.0
3	.716	66.1	.036	11.4	do.	70.5
4	.716	66.1	.039	12.3	do.	70.2
5	.718	65.5	.027	11.6	do.	70.1
6	.719	65.4	.039	10.9	do.	70.1
7	.721	64.7	.042	11.1	do.	69.9
8	.721	64.6	.032	10.7	do.	70.6
9	.723	64.1	.045	11.2	do.	69.9
10	.726	63.5	.038	10.1	do.	69.9
11	.727	63.0	.041	10.3	do.	69.6
12	.728	62.8	.032	10.6	do.	69.7
13	.728	62.8	.116	12.1	do.	69.3
14	.729	62.6	.076	10.4	do.	70.2
Average	.721	64.7	.047	11.2	do.	69.9

Premium-price gasoline

1	0.681	76.4	0.069	11.3	Neg.	76.2
2	.701	70.4	-	9.8	-	79.0
3	.711	67.6	.047	11.7	Neg.	77.2
4	.713	67.1	.049	10.9	do.	77.7
5	.719	65.3	.036	9.8	do.	74.7
6	.720	65.1	.039	10.4	do.	76.2
7	.720	65.0	.023	10.3	do.	76.1
8	.721	64.9	.031	11.1	do.	75.8
9	.721	64.7	.039	10.1	do.	76.0
10	.725	63.6	.031	10.8	-	76.0
11	.728	62.6	.045	11.4	Neg.	75.9
12	.730	62.2	.026	9.6	do.	76.0
13	.731	62.0	.076	10.5	do.	75.7
14	.736	60.7	.05	9.6	do.	75.4
Average	.718	65.5	.043	10.5	do.	76.3

Third-grade gasoline

1	0.729	62.7	0.035	9.4	Neg.	54.1
2	.729	62.6	.062	9.3	do.	54.5
3	.729	62.5	.04	8.2	do.	55.3
4	.732	61.8	.036	7.6	do.	51.2
5	.732	61.7	.035	8.1	do.	61.8
6	.732	61.7	.049	9.2	do.	57.6
7	.733	61.5	.02	7.9	do.	55.0
8	.734	61.2	.037	9.4	do.	63.9
9	.736	60.9	.044	8.5	do.	49.8
10	.736	60.9	.033	7.8	do.	54.2
11	.745	58.4	.041	2.6	do.	44.8
Average	.734	61.4	.039	8.0	do.	54.7

TABLE 1. - Motor-gasoline survey, winter 1936-37 - Continued R.I.3348
(Data for samples from different cities)

CHICAGO, ILL.

Regular-price gasoline (continued)

I.B.P.	Distillation range, °F.							Percent		Item
	Percent recovered						End point	Resid.	Loss	
	10	20	30	50	70	90				
87	113	133	152	203	264	352	405	1.0	1.9	1
85	112	133	155	212	274	349	400	1.0	2.7	2
88	119	144	171	225	272	348	393	1.0	2.8	3
89	115	137	163	219	277	349	407	1.0	2.5	4
89	119	145	174	229	281	354	395	.9	2.9	5
88	118	144	173	230	280	353	401	1.0	2.6	6
89	120	145	173	231	286	360	406	.9	2.5	7
86	120	150	178	230	283	360	405	1.0	2.9	8
87	121	150	173	234	285	355	396	.9	3.0	9
86	126	152	180	234	289	359	416	.8	2.3	10
99	130	157	185	239	299	365	407	1.0	1.5	11
88	124	150	180	228	275	347	387	1.0	2.4	12
87	120	146	174	224	275	340	386	1.0	3.0	13
86	130	158	182	228	272	344	386	1.0	3.0	14
88	121	146	173	226	272	353	399	1.0	2.6	Average

Premium-price gasoline (continued)

96	112	117	123	140	175	257	387	0.9	1.1	1
96	130	143	156	181	215	288	362	.6	1.4	2
86	113	131	154	204	255	326	383	.9	2.3	3
91	114	132	151	197	251	329	391	1.0	1.0	4
94	123	141	159	205	268	354	405	1.0	.5	5
89	123	148	172	219	266	339	387	1.0	2.0	6
86	122	146	171	219	266	339	383	1.0	2.6	7
87	122	148	177	228	277	345	385	1.0	2.9	8
94	122	149	178	230	276	333	390	1.0	1.0	9
84	120	144	170	226	281	350	415	1.0	2.0	10
85	119	146	175	233	290	363	406	1.0	2.0	11
90	128	157	186	235	279	345	387	.9	1.9	12
88	131	163	192	238	283	354	390	1.0	2.8	13
89	125	153	182	242	300	370	404	1.0	2.0	14
90	122	144	168	214	263	335	391	1.0	1.8	Average

Third-grade gasoline (continued)

97	136	165	193	243	295	360	396	1.0	2.7	1
93	139	173	202	252	302	371	411	1.0	2.2	2
100	134	160	188	239	299	374	414	1.0	1.0	3
100	145	175	204	255	305	373	422	1.0	.8	4
93	140	171	198	242	284	349	414	1.0	1.6	5
93	145	181	209	258	307	370	419	1.0	1.0	6
95	158	197	224	266	301	350	389	1.0	2.2	7
91	130	163	194	247	294	355	391	1.0	2.6	8
97	137	170	202	267	326	385	424	1.0	1.5	9
96	145	183	214	261	308	366	406	1.0	1.3	10
143	198	219	238	278	322	382	430	1.0	1.0	11
100	146	178	206	255	304	367	411	1.0	1.6	Average

TABLE 1. - Motor-gasoline survey, winter 1936-37 - Continued
(Data for samples from different cities)

NEW ORLEANS, LA.

Regular-price gasoline

Item	Gravity		Sulphur %	Reid vapor pressure, lb.	Copper- strip test	ASTM octane number
	Specific	°API				
1	0.728	63.0	0.048	9.3	Neg.	71.0
2	.729	62.7	.039	9.9	do.	71.0
3	.729	62.5	.040	11.1	do.	71.0
4	.730	62.4	.040	-	do.	69.0
5	.733	61.5	.062	9.5	do.	71.0
6	.737	60.6	.030	-	do.	70.0
7	.738	60.3	.029	12.2	do.	68.8
8	.740	59.8	.091	10.6	do.	70.0
9	.746	58.1	.052	9.8	do.	70.8
Average	.734	61.2	.048	10.3	do.	70.3

Premium-price gasoline

1	0.716	66.0	0.033	8.0	Neg.	80.7
2	.723	64.3	.050	8.0	do.	76.5
3	.730	62.3	.025	8.1	do.	78.5
4	.733	61.4	.075	8.7	do.	80.3
5	.744	58.6	.074	8.7	do.	77.2
6	.751	57.0	.050	7.4	do.	77.7
Average	.733	61.6	.051	8.2	do.	78.5

Third-grade gasoline

1	0.726	63.3	0.060	6.8	Neg.	60.8
2	.728	63.0	.020	-	do.	61.0
3	.732	61.9	.020	-	do.	58.0
4	.738	60.2	.040	8.5	do.	58.8
5	.739	60.1	.050	6.7	do.	60.8
6	.740	59.7	-	6.1	do.	60.5
7	.747	57.9	-	6.0	do.	60.0
Average	.736	60.9	.038	6.8	do.	60.0

TABLE 1. - Motor-gasoline survey, winter 1936-37 - Continued
(Data for samples from different cities)

R.I.3348

NEW ORLEANS, LA.

Regular-price gasoline (continued)

I.B.P.	Distillation range, °F.							Percent		Item
	Percent recovered						End point			
	10	20	30	50	70	90		Resid.	Loss	
92	142	174	200	243	283	347	412	1.0	1.8	1
91	133	166	194	244	287	348	391	1.0	2.0	2
96	135	165	195	245	281	350	387	.9	2.1	3
92	140	169	196	243	291	354	407	1.0	1.0	4
93	139	169	196	241	285	348	396	1.1	2.2	5
89	120	147	184	256	314	374	400	1.0	1.0	6
85	123	156	194	266	323	372	398	1.5	2.6	7
91	138	166	191	233	278	346	397	1.0	3.0	8
90	133	166	198	251	299	355	395	1.0	1.5	9
91	134	164	194	247	293	355	398	.9	1.9	Average

Premium-price gasoline (continued)

101	143	161	177	204	233	287	370	0.9	1.1	1
105	127	168	189	223	261	327	386	1.0	1.3	2
102	150	175	195	229	262	309	370	.9	1.4	3
94	140	166	193	238	276	331	381	1.0	1.8	4
95	144	170	193	234	277	340	403	1.0	1.3	5
97	148	180	200	256	303	355	396	1.0	1.4	6
99	142	170	193	231	269	325	384	1.0	1.4	Average

Third-grade gasoline (continued)

100	143	162	180	217	268	350	403	1.0	1.3	1
94	141	168	192	233	277	340	387	1.0	1.0	2
98	148	178	206	250	299	356	393	1.0	1.0	3
91	145	182	214	264	314	381	425	1.0	1.5	4
97	150	180	207	252	291	349	399	1.0	1.3	5
96	147	179	207	250	291	344	401	1.0	1.5	6
100	161	195	220	262	302	360	418	1.0	1.5	7
97	148	178	204	247	292	354	404	1.0	1.3	Average

TABLE 1. - Motor-gasoline survey, winter 1936-37 - Continued
(Data for samples from different cities)

ST. LOUIS, MO.

Regular-price gasoline

Item	Gravity		Sulphur, %	Reid vapor pressure, lb.	Copper- strip test	ASTM octane number
	Specific	°API				
1	0.711	67.5	0.076	11.6	Neg.	70.1
2	.715	66.5	.087	11.3	do.	70.2
3	.717	65.8	.040	11.6	do.	69.9
4	.723	64.2	.040	9.4	-	70.4
5	.724	63.9	.036	11.0	Neg.	70.0
6	.727	63.1	.031	10.7	do.	70.3
7	.728	62.9	.046	10.5	-	70.4
8	.728	62.9	.140	12.0	Neg.	69.8
9	.729	62.6	.023	11.1	do.	69.4
10	.730	62.4	.055	10.3	do.	69.4
11	.733	61.6	.066	10.3	do.	70.2
12	.741	59.5	.055	10.5	do.	69.7
Average	.735	61.1	.058	10.9	do.	70.0

Premium-price gasoline

1	0.709	68.1	0.027	10.7	Neg.	77.6
2	.711	67.6	.096	10.1	-	75.5
3	.714	66.6	.099	9.1	-	75.5
4	.719	65.3	.026	10.0	Neg.	76.1
5	.721	64.8	.132	8.8	do.	74.2
6	.726	63.4	.078	11.2	do.	76.3
7	.728	63.0	.077	11.7	do.	76.6
8	.728	62.9	.057	10.6	do.	76.4
9	.729	62.6	.017	5.5	-	76.0
10	.732	61.9	.031	9.7	Neg.	76.4
11	.733	61.5	.023	10.1	-	74.2
12	.738	60.3	.074	11.1	Neg.	76.0
Average	.724	64.0	.061	9.9	do.	75.9

Third-rate gasoline

1	0.717	65.8	0.039	10.1	Neg.	58.0
2	.729	62.7	-	9.1	do.	55.0
3	.729	62.5	.026	7.7	do.	53.4
4	.732	61.8	.033	7.9	do.	53.7
5	.734	61.4	.013	7.0	do.	52.4
6	.737	60.6	.017	8.8	do.	57.7
7	.743	58.9	.008	7.3	do.	53.3
8	.744	58.7	.020	7.0	do.	54.8
9	.746	58.3	.046	7.8	-	56.2
10	.746	58.1	.049	7.4	Neg.	53.8
Average	.736	60.9	.028	8.0	do.	54.8

TABLE 1. - Motor-gasoline survey, winter 1936-37 - Continued
(Data for samples from different cities)

ST. LOUIS, MO.

Regular-price gasoline (continued)

I.B.P.	Distillation range, °F.							Percent		Item
	Percent recovered						End point	Resid.	Loss	
	10	20	30	50	70	90				
88	116	132	151	199	259	345	397	0.9	2.1	1
91	115	130	146	192	257	345	399	.9	2.0	2
89	117	136	159	212	269	345	390	.9	2.7	3
96	135	159	185	230	271	340	405	1.0	2.0	4
89	122	147	176	232	283	362	403	.9	2.6	5
89	126	151	179	231	278	348	393	.9	2.5	6
90	126	152	177	237	295	367	406	1.0	3.2	7
85	123	150	179	225	273	342	380	1.0	3.3	8
89	124	154	187	245	301	371	409	.9	2.8	9
89	132	162	192	245	303	372	418	.9	3.1	10
88	132	161	191	241	287	359	395	.9	2.7	11
87	131	164	196	258	323	390	416	.9	2.6	12
89	125	150	177	229	284	357	401	.9	2.6	Average

Premium-price gasoline (continued)

91	122	139	156	192	228	306	381	.9	1.9	1
91	114	128	141	184	252	356	408	1.0	3.0	2
96	123	133	151	196	259	348	400	.9	2.1	3
95	128	153	180	229	271	319	360	1.0	2.0	4
98	128	143	162	209	272	351	402	1.0	1.5	5
89	132	163	190	231	279	346	381	.9	3.4	6
92	131	161	188	234	280	349	389	1.0	3.5	7
91	127	154	180	232	283	350	396	.9	2.2	8
116	158	176	191	216	241	292	359	1.0	2.0	9
94	133	159	185	236	289	359	416	1.0	2.0	10
98	130	162	193	250	299	367	407	1.0	2.0	11
87	123	150	179	244	314	390	417	.9	2.7	12
95	129	152	175	221	266	344	393	1.0	2.4	Average

Third-grade gasoline (continued)

90	122	141	163	219	280	353	407	.9	2.0	1
92	137	168	196	250	301	376	432	1.0	1.0	2
101	138	163	183	243	301	374	422	1.0	2.0	3
95	156	193	220	260	298	350	394	.7	2.2	4
108	153	181	205	252	300	366	412	1.0	1.8	5
96	145	178	205	253	300	362	401	.9	2.2	6
105	157	185	216	260	312	380	418	1.0	1.8	7
107	159	193	219	266	313	378	405	1.0	1.8	8
94	143	182	220	277	324	372	411	.9	1.8	9
104	152	187	218	272	327	388	427	1.0	1.5	10
99	130	196	205	255	306	370	413	.9	1.8	Average

TABLE 1. - Motor-gasoline survey, winter 1936-37 - Continued
(Data for samples from different cities)

MINNEAPOLIS, MINN.

Regular-price gasoline

Item	Gravity		Sulphur, %	Reid vapor pressure, lb.	Copper- strip test	ASTM octane number
	Specific	°API				
1	0.715	66.4	-	12.8	-	70.0
2	.716	66.2	0.059	12.6	Neg.	70.5
3	.717	65.9	.054	12.6	do.	69.4
4	.718	65.6	.053	11.5	do.	70.0
5	.721	64.7	-	11.4	-	70.0
6	.722	64.5	-	12.4	-	71.0
7	.725	63.7	.041	10.3	Neg.	70.3
8	.726	63.4	-	10.7	do.	69.9
9	.726	63.4	.044	10.0	do.	70.0
10	.726	63.3	.033	10.9	do.	70.0
11	.732	61.8	.064	9.7	do.	70.4
12	.738	60.2	.150	11.3	do.	69.5
Average	.723	64.1	.062	11.4	do.	70.1

Premium-price gasoline

1	0.676	77.7	0.058	11.2	Neg.	76.4
2	.698	71.3	-	9.8	-	80.0
3	.702	70.1	-	9.3	-	78.5
4	.710	67.9	-	11.4	-	78.0
5	.722	64.4	.026	11.0	Neg.	77.0
6	.726	63.4	-	10.5	-	77.5
7	.726	63.3	-	10.2	-	78.0
8	.735	61.1	-	8.8	Neg.	77.5
Average	.711	67.4	-	10.3	do.	77.9

Third-grade gasoline

1	0.730	62.3	-	10.3	-	55.0
2	.733	61.5	0.023	8.4	Neg.	52.0
3	.735	61.0	.023	8.5	do.	52.0
4	.738	60.2	-	7.3	-	54.5
5	.741	59.6	-	7.0	Neg.	54.0
Average	.736	60.9	-	8.3	do.	53.5

TABLE 1. - Motor-gasoline survey, winter 1936-37 - Continued
(Data for samples from different cities)

MINNEAPOLIS, MINN.

Regular-price gasoline (continued)

I.B.P.	Distillation range, °F.									Item
	Percent recovered						End	Percent		
	10	20	30	50	70	90	point	Resid.	Loss	
81	115	140	168	220	279	353	395	0.9	3.5	1
82	110	127	145	197	258	357	407	.9	2.1	2
84	109	129	151	208	271	342	392	1.0	2.7	3
85	117	139	164	220	281	358	405	1.0	2.8	4
83	122	151	181	234	280	347	390	.8	3.0	5
81	112	134	160	224	297	373	409	.8	2.8	6
88	126	153	182	236	289	359	409	1.0	1.7	7
91	125	151	179	232	282	347	390	.9	2.5	8
87	125	151	180	235	283	360	408	.9	2.4	9
88	123	151	181	234	291	364	407	.9	2.7	10
94	135	163	190	239	289	355	395	1.0	2.3	11
87	126	157	186	237	290	361	401	1.0	3.6	12
86	120	146	172	226	283	356	401	.9	2.7	Average

Premium-price gasoline (continued)

96	110	117	120	136	169	245	359	1.5	1.5	1
89	119	132	146	180	226	291	350	.8	1.2	2
93	131	144	156	181	214	287	362	.9	2.1	3
85	114	132	151	196	248	315	374	.8	2.0	4
85	115	140	170	224	272	322	364	1.4	2.6	5
86	119	138	160	201	248	317	370	.8	2.0	6
87	131	160	188	233	269	317	356	.6	2.4	7
88	130	160	191	248	304	370	419	1.0	2.0	8
89	121	140	160	200	244	308	369	1.0	2.0	Average

Third-grade gasoline (continued)

87	137	171	201	253	309	382	416	0.9	3.0	1
95	143	175	202	252	310	374	413	1.1	1.9	2
95	143	177	206	258	310	378	422	1.0	1.9	3
95	150	187	215	258	298	360	420	.9	1.6	4
98	149	181	208	258	308	377	426	1.0	1.0	5
94	144	178	206	256	307	354	419	1.0	1.9	Average

TABLE 1.- Motor-gasoline survey, winter 1936-37 - Continued
(Data for samples from different cities)

OMAHA, NEBR.

KANSAS CITY, MO.

Regular-price gasoline

Item	Gravity		Sulphur, %	Reid vapor pressure, lb.	Copper strip test	ASTM octane number
	Spe- cific	°API				
1	0.711	67.5	0.046	11.6	Neg.	70.1
2	.721	64.9	.049	11.4	do.	69.2
3	.721	64.7	.035	11.7	do.	70.0
4	.722	64.5	.033	9.5	do.	70.4
5	.723	64.2	.033	10.6	do.	70.5
6	.723	64.1	-	12.2	-	70.0
7	.723	64.1	.037	10.0	Neg.	70.0
8	.725	63.8	-	10.8	-	70.0
9	.725	63.8	.030	10.1	Neg.	69.9
10	.725	63.6	.033	10.7	do.	70.1
11	.727	63.1	.050	10.8	do.	69.5
12	.729	62.6	-	11.4	-	70.0
13	.731	62.0	.030	8.7	Neg.	68.4
Average	.723	64.1	.038	10.7	do.	69.9

Premium-price gasoline

1	0.699	71.0	0.030	10.9	Neg.	79.1
2	.703	69.9	.020	9.5	do.	78.7
3	.710	67.8	.036	9.4	-	75.0
4	.715	66.3	.033	11.5	Neg.	78.5
5	.723	64.2	.025	8.6	do.	74.6
6	.730	62.3	.044	8.1	do.	76.0
7	.730	62.0	.033	7.5	do.	76.9
8	.732	61.7	-	10.5	-	76.0
9	.734	61.2	.030	8.4	Neg.	77.8
Average	.719	65.2	.031	9.4	do.	77.0

Third-grade gasoline

1	0.726	63.5	-	8.0	-	56.0
2	.730	62.2	0.031	7.0	Neg.	52.0
3	.731	62.1	.053	7.2	do.	52.0
4	.733	61.5	-	7.7	-	55.0
5	.734	61.4	.033	6.1	Neg.	49.0
6	.735	61.0	-	7.6	-	50.3
Average	.731	62.0	.039	7.3	Neg.	52.4

TABLE 1.- Motor-gasoline survey, winter 1936-37 - Continued
(Data for samples from different cities)

OMAHA, NEBR.
KANSAS CITY, MO.

Regular-price gasoline (continued)

I.B.P.	Distillation range, °F.							Percent		Item
	Percent recovered						End point			
	10	20	30	50	70	90		Resid.	Loss	
87	111	128	148	199	260	355	410	1.1	1.9	1
85	120	146	173	219	260	316	360	1.0	2.6	2
84	115	136	159	208	260	340	393	.9	2.7	3
86	128	158	187	239	281	366	374	.9	2.1	4
87	122	147	176	249	281	348	387	.9	2.3	5
80	115	144	176	238	301	374	412	.8	3.2	6
88	123	148	176	233	287	359	401	.9	2.4	7
84	125	154	183	232	280	358	402	.7	2.5	8
89	125	149	180	236	291	361	417	1.1	1.9	9
88	125	153	181	232	280	344	393	.9	2.0	10
94	126	150	178	233	287	357	413	1.0	3.0	11
82	122	154	188	248	302	372	410	.7	2.8	12
97	134	164	193	246	297	371	426	1.1	2.0	13
87	122	149	177	232	282	328	400	.9	2.4	Average

Premium-price gasoline (continued)

87	122	134	148	183	228	289	355	1.0	1.4	1
99	133	145	156	181	213	287	366	.9	1.6	2
96	121	133	147	185	239	346	413	.9	1.3	3
83	114	136	159	211	265	325	368	1.1	2.1	4
95	128	154	181	231	275	336	395	1.0	1.5	5
88	128	155	187	241	294	363	405	1.1	1.7	6
92	135	162	189	235	283	342	394	1.0	1.0	7
84	126	159	193	249	304	370	406	.5	2.5	8
96	140	173	200	246	278	322	358	1.0	1.0	9
91	127	150	173	218	264	331	384	.9	1.6	Average

Third-grade gasoline (continued)

92	142	169	193	231	270	332	402	.7	1.4	1
100	145	172	200	250	305	312	420	1.1	1.4	2
96	140	170	200	252	308	376	419	1.0	2.0	3
96	146	177	205	253	297	356	407	.8	3.0	4
103	130	145	175	228	279	338	416	1.0	2.0	5
94	143	176	205	258	309	374	421	.8	1.3	6
97	141	168	196	245	295	348	414	.9	1.9	Average

TABLE 1.- Motor-gasoline survey, winter 1936-37 - Continued
(Data for samples from different cities)

TULSA, OKLA.

Item	Gravity		Sulphur, %	Reid vapor pressure, lb.	Copper- strip test	ASTM octane number
	Specific	°API				
1	0.709	68.2	-	12.0	Neg.	71.0
2	.712	67.2	0.019	10.9	do.	69.6
3	.716	66.1	-	12.3	-	70.0
4	.721	64.9	.054	10.0	Neg.	69.3
5	.721	64.7	.025	10.4	do.	70.2
6	.724	63.9	.031	10.4	do.	69.8
7	.725	63.6	.030	10.1	do.	69.5
8	.727	63.2	.015	10.9	do.	70.0
9	.727	63.2	.027	8.7	do.	70.1
10	.729	62.7	.028	10.5	do.	69.8
11	.729	62.7	.035	10.3	do.	70.3
12	.729	62.5	-	9.3	do.	70.5
13	.730	62.4	.019	11.0	do.	69.3
14	.732	61.7	.059	8.9	do.	70.0
15	.732	61.7	.082	10.0	do.	69.6
16	.734	61.3	.015	9.2	do.	68.9
Average	.725	63.8	.034	10.3	do.	69.9

Premium-price gasoline						
1	0.719	65.4	-	8.0	Neg.	75.5
2	.720	65.0	-	8.6	-	77.0
3	.722	64.4	-	8.7	Neg.	76.4
4	.726	63.4	0.066	8.6	do.	75.3
5	.727	63.0	.022	10.5	do.	76.8
6	.728	62.9	-	10.7	-	77.0
7	.730	62.3	.030	10.0	Neg.	77.0
8	.735	61.0	-	8.7	do.	76.5
9	.738	60.3	-	8.4	do.	77.5
10	.741	59.4	.040	6.0	do.	70.0
Average	.729	62.7	.040	8.8	do.	75.9

Third-grade gasoline						
1	0.717	65.8	0.021	9.3	Neg.	54.1
2	.729	62.7	.023	9.2	do.	54.6
3	.729	62.6	.016	7.0	do.	57.8
4	.729	62.5	.026	7.9	do.	50.4
5	.730	62.4	.025	7.4	do.	50.4
6	.731	62.2	.030	8.2	do.	53.4
7	.732	61.9	.028	7.2	do.	51.8
8	.736	60.7	.005	5.7	do.	50.0
9	.740	59.6	.027	6.6	do.	54.0
10	.741	59.4	-	9.3	-	62.7
Average	.731	62.0	.022	7.8	Neg.	53.9

TABLE 1.- Motor-gasoline survey, winter 1936-37 - Continued
(Data for samples from different cities)

TULSA, OKLA.

Regular-price gasoline (continued)

I.B.P.	Distillation range, °F.							Percent		Item
	Percent recovered						End point			
	10	20	30	50	70	90		Resid.	Loss	
88	115	130	150	198	250	322	378	1.0	2.0	1
90	116	132	149	197	264	365	422	1.0	1.8	2
85	120	143	170	223	282	355	407	.7	3.5	3
89	118	138	159	210	269	341	388	.9	2.0	4
91	126	158	184	235	283	356	396	1.0	3.0	5
87	121	147	177	230	281	348	407	1.1	1.7	6
88	124	153	184	241	296	369	403	1.0	3.1	7
85	126	158	189	242	301	369	408	.8	3.0	8
93	135	166	194	247	293	354	394	1.0	2.5	9
89	130	160	191	245	302	370	420	1.0	3.0	10
89	128	159	187	238	284	352	396	1.0	2.5	11
91	137	166	194	244	289	352	392	1.0	2.1	12
92	123	152	183	243	295	359	403	1.0	1.0	13
90	135	166	197	246	299	358	391	1.0	3.0	14
92	129	155	182	229	276	351	414	1.0	2.1	15
92	129	156	185	242	302	371	408	.9	2.7	16
89	126	152	173	232	285	356	402	1.1	2.4	Average

Premium-price gasoline (continued)

95	126	142	157	198	257	350	426	1.0	1.0	1
97	126	144	160	200	251	320	377	.9	1.1	2
96	124	140	158	206	268	364	420	.9	1.6	3
94	131	158	181	225	271	349	404	1.2	1.8	4
89	128	156	183	238	291	365	421	1.3	2.6	5
86	126	155	189	237	286	351	406	.8	2.8	6
100	136	163	180	237	298	354	401	1.2	2.4	7
93	136	166	194	242	290	355	400	1.0	2.5	8
94	138	172	203	259	312	371	406	1.0	2.0	9
102	140	165	200	252	300	366	411	1.0	2.0	10
95	131	156	181	229	282	355	407	1.0	2.0	Average

Third-grade gasoline (continued)

92	123	142	162	216	287	370	420	1.0	1.6	1
92	131	163	195	251	305	376	430	1.1	2.0	2
98	148	177	201	243	288	350	390	.8	2.5	3
94	138	170	201	254	301	367	422	1.0	1.5	4
100	143	175	203	252	298	363	414	1.0	1.5	5
92	139	174	202	252	303	372	427	1.0	2.0	6
96	140	172	203	250	303	372	435	1.1	1.8	7
98	148	177	201	243	288	350	390	.8	2.5	8
97	156	191	216	260	304	365	422	1.0	1.6	9
90	131	156	179	229	290	365	410	.8	1.0	10
95	140	170	196	245	297	365	416	1.0	1.8	Average

TABLE 1.- Motor-gasoline survey, winter 1936-37 - Continued
(Data for samples from different cities)

NORTH TEXAS

Regular-price gasoline

Item	Gravity		Sulphur, %	Reid vapor pressure, lb.	Copper- strip test	ASTM octane number
	Specific	°API				
1	0.726	63.4	0.081	10.4	Neg.	69.0
2	.729	62.6	.073	9.8	do.	69.1
3	.731	62.0	.037	9.1	do.	70.1
4	.734	61.2	.054	8.8	do.	67.7
5	.735	61.1	.042	8.6	do.	68.8
6	.736	60.8	.016	8.7	do.	70.4
7	.736	60.8	.044	9.2	do.	68.7
8	.737	60.6	.053	8.5	do.	69.7
9	.737	60.5	.050	8.8	do.	69.2
Average	.734	61.4	.050	9.1	do.	69.2

Premium-price gasoline

1	0.721	64.7	0.028	9.1	Neg.	77.6
2	.727	63.1	.052	8.5	do.	74.6
3	.735	61.1	.012	8.3	do.	76.3
4	.735	61.0	.046	8.3	do.	76.9
5	.736	60.8	.062	7.8	do.	76.1
6	.742	59.3	.029	8.7	do.	75.6
7	.743	58.8	.029	8.5	do.	75.5
Average	.734	61.3	.037	8.5	do.	76.1

Third-grade gasoline

1	0.725	63.7	0.055	8.8	Neg.	53.6
2	.732	61.7	.028	8.8	do.	52.4
3	.733	61.4	.023	7.6	do.	54.0
4	.737	60.6	.016	7.5	do.	53.8
5	.738	60.3	.036	7.5	do.	59.8
6	.738	60.2	.063	4.9	do.	53.8
7	.739	60.0	.039	8.0	do.	58.7
Average	.735	61.1	.037	7.6	do.	55.2

TABLE 1.- Motor-gasoline survey, winter 1936-37 - Continued
(Data for samples from different cities)

NORTH TEXAS

Regular-price gasoline (continued)

I.B.P.	Distillation range, °F.							Percent		Item
	Percent recovered						End			
	10	20	30	50	70	90	point	Resid.	Loss	
90	133	165	193	240	284	345	381	1.0	2.9	1
92	142	172	196	242	288	354	395	1.0	2.5	2
94	140	167	194	243	288	353	405	1.0	1.6	3
93	138	171	200	251	299	352	393	.9	1.7	4
93	143	174	202	249	294	354	400	1.0	1.6	5
88	119	151	183	239	287	345	396	1.0	1.5	6
94	145	177	204	253	301	362	427	1.2	3.7	7
91	137	170	200	252	301	365	406	.9	1.7	8
92	144	177	205	252	298	357	398	1.0	1.8	9
92	138	169	197	247	293	354	400	1.0	2.1	Average

Premium-price gasoline (continued)

98	130	156	184	232	272	336	381	1.0	1.5	1
86	125	153	180	232	281	343	392	1.0	1.0	2
88	139	173	201	245	288	346	397	1.0	1.0	3
88	135	165	193	240	283	337	396	1.0	1.5	4
90	135	170	197	243	284	339	393	1.0	1.5	5
93	139	171	203	252	294	349	398	1.0	1.0	6
84	131	166	198	247	291	344	400	1.0	1.0	7
90	133	165	194	242	285	342	394	1.0	1.2	Average

Third-grade gasoline (continued)

90	135	164	191	237	277	342	389	1.0	1.5	1
96	138	171	200	249	291	361	410	1.0	1.5	2
92	149	187	216	253	288	335	396	1.0	1.5	3
98	158	192	221	260	290	348	410	1.0	1.5	4
94	145	180	213	255	298	357	414	1.0	1.5	5
94	153	192	217	252	287	327	383	1.0	1.0	6
97	149	187	216	260	301	354	421	1.0	1.5	7
94	147	182	211	252	290	346	403	1.0	1.4	Average

TABLE 1.- Motor-gasoline survey, winter 1936-37 - Continued
 (Data for samples from different cities)

SOUTH TEXAS

Regular-price gasoline

Item	Gravity		Sulphur, %	Reid vapor pressure, lb.	Copper- strip test	ASTM octane number
	Specific	°API				
1	0.728	63.0	0.031	8.9	Neg.	69.6
2	.728	63.0	.067	9.1	do.	69.3
3	.729	62.7	.038	10.0	do.	70.6
4	.731	62.0	.024	8.2	do.	64.0
5	.740	59.9	.063	8.2	do.	69.7
6	.740	59.8	.068	8.5	do.	69.6
7	.740	59.7	.075	9.4	do.	69.6
8	.742	59.2	.081	8.5	do.	69.8
9	.754	56.1	.056	7.6	do.	70.6
Average	.737	60.6	.056	8.7	do.	69.2

Premium-price gasoline

1	0.726	63.4	0.028	8.5	Neg.	76.5
2	.727	63.1	.044	7.3	do.	79.0
3	.732	61.9	.044	8.7	do.	78.7
4	.732	61.7	.043	8.7	do.	79.2
5	.733	61.5	.046	7.3	do.	75.9
6	.742	59.3	.038	6.5	do.	75.5
7	.746	58.2	.060	6.7	do.	75.8
Average	.734	61.3	.043	7.7	do.	77.2

Third-grade gasoline

1	0.730	62.4	0.022	9.7	Neg.	57.8
2	.734	61.2	.040	5.4	do.	54.1
3	.736	60.7	.034	7.2	do.	57.0
4	.737	60.5	.039	5.8	do.	55.5
5	.737	60.4	.044	7.3	do.	57.7
6	.738	60.2	.043	7.5	do.	57.6
7	.741	59.6	.034	4.6	do.	50.0
8	.747	58.0	.083	5.8	do.	56.8
Average	.737	60.4	.042	6.7	do.	55.8

TABLE 1.- Motor-gasoline survey, winter 1936-37 - Continued
(Data for samples from different cities)

SOUTH TEXAS

Regular-price gasoline (continued)

I.B.P.	Distillation range, °F.							Percent		Item
	Percent recovered						End point			
	10	20	30	50	70	90		Resid.	Loss	
86	135	163	190	237	278	343	388	1.0	1.5	1
93	132	154	174	214	254	326	390	.9	1.7	2
90	133	162	192	242	287	346	392	.8	2.2	3
89	133	162	192	237	278	334	372	1.0	1.5	4
93	142	173	202	251	296	352	397	.9	1.6	5
96	141	172	200	248	291	350	399	.9	1.2	6
90	138	171	205	257	300	356	419	1.1	2.3	7
95	141	170	199	249	296	353	390	.8	1.6	8
100	156	161	221	269	312	357	401	1.0	1.5	9
92	139	165	197	245	288	347	395	.9	1.7	Average

Premium-price gasoline (continued)

89	137	163	185	223	263	320	372	1.0	1.0	1
97	140	157	174	203	233	288	370	1.3	1.0	2
94	135	162	188	233	271	326	378	1.0	1.3	3
93	138	166	192	233	271	331	381	.9	1.9	4
96	145	172	199	247	289	339	378	.9	1.1	5
103	151	178	203	245	288	345	397	1.0	1.0	6
97	146	174	202	250	297	353	399	.9	.9	7
96	142	167	192	233	273	329	382	1.0	1.2	Average

Third-grade gasoline (continued)

86	135	167	197	244	290	348	388	1.0	2.0	1
98	147	181	209	249	288	350	398	1.0	1.0	2
91	146	181	212	256	293	331	384	1.0	1.0	3
98	151	182	207	248	287	342	426	1.0	.5	4
94	146	178	208	251	294	350	398	1.0	1.0	5
92	137	170	205	255	297	348	394	1.0	1.0	6
108	167	196	220	257	299	360	407	1.0	.5	7
96	152	184	219	256	300	360	419	1.0	1.0	8
95	148	180	210	252	294	349	402	1.0	1.0	Average

TABLE 1.- Motor-gasoline survey, winter 1936-37 - Continued
(Data for samples from different cities)

DENVER, COLO.

Regular-price gasoline

Item	Gravity		Sulphur, %	Reid vapor pressure, lb.	Copper- strip test	ASTM octane number
	Specific	°API				
1	0.711	67.6	0.019	11.1	Neg.	69.2
2	.721	64.8	.051	11.0	do.	68.9
3	.722	64.4	.045	11.7	do.	69.8
4	.724	64.0	.033	10.6	-	70.3
5	.724	63.9	.054	8.6	Neg.	69.3
6	.725	63.7	-	11.2	-	70.0
7	.727	63.2	-	10.4	-	69.3
8	.728	62.9	.030	9.9	Neg.	71.1
9	.734	61.3	.032	10.2	do.	69.8
10	.734	61.2	.036	10.3	do.	69.4
11	.735	61.0	.010	8.6	do.	69.4
12	.737	60.6	.036	10.9	do.	71.2
Average	.727	63.2	.035	10.4	do.	69.8

Premium-price gasoline

1	0.707	68.6	0.031	7.9	-	77.0
2	.711	67.6	.013	11.2	Neg.	76.6
3	.728	62.9	.036	9.6	do.	77.4
4	.732	61.8	.031	9.4	do.	74.6
5	.733	61.6	.030	9.3	-	76.5
6	.733	61.5	.021	7.0	-	78.0
7	.736	60.8	.027	8.7	Neg.	74.8
Average	.725	63.6	.027	9.0	do.	76.4

Third-grade gasoline

1	0.724	64.0	0.017	8.9	Neg.	53.5
2	.728	62.8	.055	7.5	do.	51.5
3	.729	62.6	.030	9.4	-	55.9
4	.733	61.6	-	8.5	Neg.	54.5
5	.733	61.4	.024	8.4	do.	52.7
6	.736	60.7	.016	8.4	-	50.2
7	.742	59.3	.029	10.0	Neg.	64.0
8	.745	58.4	.020	7.7	do.	54.0
9	.746	58.2	.022	7.9	do.	51.0
10	.747	57.9	.024	8.5	do.	56.4
Average	.736	60.7	.026	8.5	do.	54.4

TABLE 1.- Motor-gasoline survey, winter 1936-37 - Continued
(Data for samples from different cities)

DENVER, COLO

Regular-price gasoline (continued)

I.B.P.	Distillation range, °F.						End point	Percent		Item
	Percent recovered							Resid.	Loss	
	10	20	30	50	70	90				
89	115	131	148	197	266	366	415	0.7	1.8	1
85	113	135	159	216	278	357	401	.9	2.3	2
87	125	152	181	231	279	353	399	.9	2.9	3
87	127	153	181	236	286	350	390	1.0	2.6	4
84	124	150	177	227	274	349	410	.9	1.9	5
86	131	160	189	242	293	367	407	.7	3.8	6
91	126	156	187	245	302	370	409	.8	2.1	7
88	127	156	183	229	274	342	387	.9	2.4	8
92	140	192	200	249	297	367	404	.9	2.9	9
88	131	164	196	256	314	382	415	.9	3.2	10
94	135	168	197	247	295	365	402	1.0	2.0	11
89	138	170	200	248	292	352	390	.9	3.4	12
88	128	157	183	235	288	360	402	.9	2.6	Average

Premium-price gasoline (continued)

100	133	146	159	196	237	300	369	0.9	1.5	1
87	117	136	155	202	253	323	376	.9	1.7	2
92	133	161	191	242	286	344	388	.9	1.8	3
89	128	158	188	242	298	368	412	1.0	1.7	4
91	131	160	188	240	292	359	403	.9	2.3	5
104	138	156	175	223	281	359	426	.9	1.0	6
96	137	168	195	247	299	364	409	1.0	1.4	7
94	131	155	179	227	278	345	398	.9	1.6	Average

Third-grade gasoline (continued)

85	130	164	190	228	270	328	383	1.0	2.5	1
87	126	161	191	240	296	369	408	1.0	3.0	2
88	137	170	200	250	298	366	410	1.0	3.5	3
92	141	175	204	253	307	371	410	1.0	1.0	4
92	144	177	204	253	303	367	403	1.0	2.2	5
94	151	185	212	260	313	381	419	.9	2.3	6
90	147	183	209	256	303	362	400	.9	3.6	7
99	159	195	224	271	319	379	419	.9	2.2	8
96	157	197	225	273	322	385	422	1.0	2.3	9
92	141	180	217	280	331	386	424	.9	1.8	10
92	143	179	208	256	306	369	410	1.0	2.4	Average

TABLE 1.- Motor-gasoline survey, winter 1936-37 - Continued
 (Data for samples from different cities)

SALT LAKE CITY, UTAH

Regular-price gasoline

Item	Gravity		Sulphur, %	Reid vapor pressure lb.	Copper- strip test	ASTM octane number
	Specific	°API				
1	0.727	63.1	0.042	10.9	Neg.	70.0
2	.729	62.5	.035	11.5	do.	70.2
3	.729	62.5	.043	11.3	do.	70.2
4	.729	62.5	.041	11.3	do.	70.3
5	.732	61.9	.026	9.9	do.	70.0
6	.735	61.0	.032	8.7	do.	69.4
7	.737	60.4	.057	9.6	do.	69.7
8	.750	57.2	.089	11.7	do.	70.2
Average	.733	61.4	.046	10.6	do.	70.0

Third-grade gasoline

1	0.744	58.7	0.060	10.4	Neg.	62.9
2	.746	58.2	.025	9.5	do.	59.0
3	.746	58.2	.029	9.7	do.	57.8
4	.746	58.2	.032	7.4	do.	58.0
5	.746	58.2	.039	8.0	do.	57.5
6	.746	58.1	.035	7.3	do.	57.8
7	.747	57.9	.020	7.9	do.	56.8
8	.751	57.0	.032	7.4	do.	54.9
9	.752	56.6	.074	7.8	do.	59.3
Average	.747	57.9	.038	8.4	do.	58.2

TABLE 1.- Motor-gasoline survey, winter 1936-37 - Continued
(Data for samples from different cities)

SALT LAKE CITY, UTAH

Regular-price gasoline (continued)

I.B.P.	Distillation range, °F.						End point	Percent		Item
	Percent recovered							Resid.	Loss	
	10	20	30	50	70	90				
72	109	140	174	235	298	372	388	1.0	5.0	1
81	114	144	177	240	296	366	397	1.0	3.8	2
77	113	142	175	238	295	367	397	1.0	4.0	3
67	107	139	173	236	291	361	338	1.0	4.0	4
77	122	152	183	238	286	360	389	1.0	3.5	5
83	126	157	189	245	299	368	403	1.0	3.3	6
82	116	151	186	241	290	339	383	1.0	3.0	7
81	119	142	163	195	257	379	425	1.0	3.5	8
78	116	146	178	234	289	364	434	1.0	3.8	Average

Third-grade gasoline (continued)

84	129	162	192	246	317	405	420	1.0	5.0	1
84	119	152	186	254	314	378	412	1.0	1.0	2
78	122	161	196	260	324	384	423	1.0	2.5	3
79	120	156	192	256	326	384	412	1.0	2.5	4
82	122	158	190	259	328	380	412	1.0	2.5	5
76	123	159	194	258	323	381	413	1.0	2.0	6
82	123	160	194	258	327	381	414	1.0	1.5	7
89	148	185	215	264	301	362	401	1.0	1.5	8
81	129	170	207	261	311	364	399	1.0	1.5	9
82	126	163	196	257	319	380	412	1.0	2.2	Average

TABLE 1.- Motor-gasoline survey, winter 1936-37 - Continued
(Data for samples from different cities)

SEATTLE, WASH.

Regular-price gasoline

Item	Gravity		Sulphur, %	Reid vapor pressure, lb.	Copper- strip test	ASTM octane number
	Specific	°API				
A	0.737	60.4	0.09	9.1	Neg.	70.5
B	.742	59.2	.04	9.8	do.	71.5
C	.741	59.4	.14	9.6	do.	72.0
D	.740	59.8	.30	10.2	do.	71.0
E	.730	62.4	.14	9.8	do.	71.0
F	.737	60.6	.15	9.0	do.	71.5
G	.732	61.7	.07	10.6	do.	72.0
Average	.737	60.5	.13	9.7	do.	71.4

Premium-price gasoline

A	0.730	62.3	0.01	6.7	Neg.	81.5
B	.740	59.7	.03	9.4	do.	80.5
C	.730	62.4	.03	8.7	do.	79.5
D	.737	60.6	.03	9.2	do.	81.0
E	.729	62.7	.02	8.0	do.	82.5
F	.737	60.4	.10	9.1	do.	77.0
G	.735	61.1	.06	8.3	do.	82.5
Average	.734	61.3	.04	8.5	do.	80.5

Third-grade gasoline

A	0.752	56.7	0.14	8.0	Neg.	60.5
B	.754	56.2	.12	9.6	do.	63.5
C	.753	56.3	.07	7.9	do.	61.5
D	.746	58.1	.19	9.6	do.	63.5
E	.745	58.4	.07	8.2	do.	62.5
F	.752	56.7	.23	7.7	do.	63.5
G	.729	62.6	.04	9.8	do.	65.0
Average	.747	57.9	.12	8.7	do.	62.9

TABLE 1.- Motor-gasoline survey, winter, 1936-37 - Continued
(Data for samples from different cities)

SEATTLE, WASH.

Regular-price gasoline (continued)

I.B.P.	Distillation range, °F.									Item
	Percent recovered						End point	Percent		
	10	20	30	50	70	90		Resid.	Loss	
88	130	159	191	251	300	360	406	1.0	2	A
85	128	160	194	245	298	369	407	1.0	3	B
87	134	166	200	251	300	363	400	1.0	3	C
86	127	158	191	250	298	362	397	1.0	3.5	D
87	120	145	171	232	281	344	391	1.0	2	E
88	133	164	193	241	284	344	396	1.0	2	F
84	120	150	177	226	273	360	405	1.0	4	G
86	127	157	188	242	291	357	400	1.0	2.8	Average

Premium-price gasoline (continued)

104	147	167	184	212	240	290	347	1.0	1	A
90	134	163	191	239	284	351	398	1.0	2	B
93	131	151	174	214	249	301	371	1.0	1	C
91	134	164	190	234	274	339	394	1.0	2	D
100	134	154	174	221	282	310	366	1.0	1	E
94	136	165	192	241	280	337	386	1.0	1.5	F
91	137	163	186	222	260	337	405	1.0	1.5	G
95	136	161	184	226	267	324	381	1.0	1.4	Average

Third-grade gasoline (continued)

87	138	179	218	277	334	388	420	1.0	2	A
88	140	184	224	280	323	380	413	1.0	2.5	B
98	152	190	220	268	317	380	422	1.0	1.5	C
87	140	181	220	274	321	382	422	1.0	3.0	D
89	139	170	204	254	306	376	424	1.0	2	E
98	151	190	219	268	310	382	433	1.0	2	F
90	127	154	184	234	286	366	414	1.0	3	G
91	141	178	213	265	314	379	421	1.0	2.3	Average

TABLE 1.- Motor-gasoline survey, winter, 1936-37 - Continued
(Data for samples from different cities)

LOS ANGELES, CALIF.

Regular-price gasoline

Item	Gravity		Sulphur, %	Reid vapor pressure, lb.	Copper- strip test	ASTM octane number
	Specific	°API				
1	0.736	60.7	0.17	-	Neg.	71
2	.742	59.3	.35	-	do.	71
3	.745	58.5	.08	-	do.	70
4	.745	58.5	.04	-	do.	70
5	.744	58.7	.15	-	do.	71
6	.742	59.1	.16	-	do.	70
7	.736	60.7	.06	-	do.	71
8	.746	58.1	.07	-	do.	70
9	.753	56.4	.04	-	do.	71
10	.738	60.3	.14	-	do.	71
11	.745	58.3	.04	-	do.	71
12	.741	59.5	.21	-	do.	71
13	.750	57.2	.08	-	do.	69
Average	.743	58.9	.122	-	do.	70.5

Premium-price gasoline

1	0.729	62.6	0.03	-	Neg.	81
2	.731	62.0	.04	-	do.	81
3	.733	61.5	.03	-	do.	80
4	.742	59.3	.04	-	do.	79
5	.727	63.2	.03	-	do.	81
6	.739	60.0	.06	-	do.	80
7	.736	60.8	.06	-	do.	80
8	.747	57.9	.05	-	do.	78
9	.742	59.3	.07	-	do.	80
10	.731	62.1	.04	-	do.	80
11	.748	57.7	.03	-	do.	79
12	.732	61.8	.06	-	do.	81
13	.743	58.8	.07	-	do.	79
Average	.737	60.5	.047	-	do.	80

Third-grade gasoline

1	0.747	58.0	0.13	-	Neg.	62
2	.748	57.8	.14	-	do.	63
3	.754	56.3	.09	-	do.	63
4	.754	56.4	.08	-	do.	64
5	.754	56.3	.09	-	do.	62
6	.747	58.0	.27	-	do.	61
7	.740	59.8	.07	-	do.	62
8	.747	57.9	.03	-	do.	60
9	.752	56.7	.06	-	do.	64
10	.746	58.2	.10	-	do.	63
11	.759	55.0	.11	-	do.	63
12	.756	55.7	.09	-	do.	60
13	.752	56.7	.06	-	do.	62
Average	.750	57.1	.102	-	do.	62.2

TABLE 1.- Motor-gasoline survey, winter, 1936-37 - Continued
(Data for samples from different cities)

LOS ANGELES, CALIF.

Regular-price gasoline (continued)

I.B.P.	Distillation range, °F.						End point	Percent		Item
	Percent recovered							Resid.	Loss	
	10	20	30	50	70	90				
92	135	158	193	244	292	352	400	1.0	-	1
88	132	163	198	258	302	353	391	.8	-	2
96	142	185	215	257	298	354	399	.8	-	3
88	135	169	204	249	294	362	402	.8	-	4
91	134	168	201	251	297	360	397	.8	-	5
93	137	168	197	250	290	350	395	.8	-	6
86	128	158	186	232	281	365	407	.7	-	7
92	149	186	213	259	307	362	397	.7	-	8
92	141	181	219	271	317	371	404	.8	-	9
92	132	160	190	241	289	346	396	.7	-	10
88	136	166	199	250	293	360	405	.8	-	11
88	146	173	206	252	297	367	410	.8	-	12
100	159	191	218	258	297	356	399	.7	-	13
91	139	171	203	252	296	358	400	.8	-	Average

Premium-price gasoline (continued)

95	134	152	176	223	261	314	368	0.8	-	1
90	139	168	188	225	263	316	375	.8	-	2
108	154	172	189	214	247	292	349	.7	-	3
97	145	172	202	248	288	346	376	.7	-	4
100	131	154	176	214	248	297	355	.8	-	5
97	141	171	198	243	283	340	381	.8	-	6
97	149	171	193	227	263	328	402	.8	-	7
97	141	180	210	255	298	356	394	.8	-	8
98	145	171	201	246	286	340	389	.6	-	9
96	134	155	177	223	267	315	377	1.0	-	10
93	150	181	210	253	293	348	390	.7	-	11
93	143	170	192	223	258	333	397	1.0	-	12
99	154	184	209	243	279	332	386	.9	-	13
97	143	169	194	234	272	327	380	.8	-	Average

Third-grade gasoline (continued)

94	138	173	212	267	319	386	425	1.0	-	1
91	147	195	230	282	328	389	417	.8	-	2
89	141	186	226	279	323	381	412	.9	-	3
87	141	191	231	281	327	386	414	.9	-	4
98	156	192	223	273	312	387	424	.7	-	5
99	151	178	221	271	311	368	409	.7	-	6
95	140	171	200	244	287	354	416	.8	-	7
95	151	189	221	262	306	360	401	.7	-	8
93	151	187	224	272	326	388	419	1.0	-	9
94	137	175	208	267	325	389	423	.8	-	10
96	154	199	235	283	328	384	416	.8	-	11
95	167	212	240	283	314	383	423	.9	-	12
98	152	190	224	277	327	391	419	.9	-	13
94	148	188	223	272	318	380	417	.8	-	Average

TABLE 1.- Motor-gasoline survey, winter, 1936-37 - Continued

(Data for samples from different cities)

SAN FRANCISCO BAY REGION, CALIF

Regular-price gasoline

Item	Gravity		Sulphur %	Reid vapor pressure, lb.	Copper- strip test	ASTM octane number
	Specific	°API				
A	0.742	59.1	0.13	9.5	Neg.	70.0
B	.744	58.7	.042	10.0	do.	70.5
C	.748	57.6	.210	9.0	do.	71.0
D	.744	58.6	.290	9.3	do.	70.5
E	.739	60.1	.180	9.1	do.	70.5
F	.738	60.3	.170	9.1	do.	70.5
G	.747	57.8	.052	8.2	do.	71.0
H	.737	60.6	.072	9.5	do.	71.5
I	.744	58.7	.037	8.0	do.	70.5
J(1)	.743	59.0	.190	8.9	do.	71.0
(2)	.757	55.8	.052	10.5	do.	67.0
(3)	.764	53.8	.250	7.0	do.	66.5
(4)	.754	56.1	.160	8.5	do.	67.0
J(ave.)	.754	56.2	.163	8.7	do.	67.9
K(1)	.736	60.9	.180	9.9	do.	70.5
(2)	.748	57.7	.200	9.3	do.	70.5
K(ave.)	.742	59.3	.190	9.6	do.	70.5
L(1)	.740	59.8	.087	9.0	do.	71.0
(2)	.741	59.6	.110	9.4	do.	71.0
L(ave.)	.740	59.7	.099	9.2	do.	71.0
Average	.743	58.9	.151	9.1	do.	70.5

Premium-price gasoline

A	0.730	62.4	0.027	7.0	Neg.	80.0
B	.742	59.2	.058	9.4	do.	78.5
C	.729	62.7	.044	7.7	do.	80.0
D	.731	62.0	.046	7.3	do.	81.0
E	.728	62.8	.031	8.9	do.	81.0
F	.729	62.7	.051	8.6	do.	81.0
G	.741	59.5	.053	9.5	do.	79.0
H	.740	59.7	.076	8.0	do.	81.0
I	.739	60.1	.032	8.7	do.	79.0
J	.729	62.6	.037	8.1	do.	80.0
K	.730	62.3	.035	8.1	do.	81.0
L	.742	59.3	.043	7.3	do.	79.5
Average	.734	61.3	.044	8.2	do.	80.0

TABLE 1.- Motor-gasoline survey, winter 1936-37 - Continued

(Data for samples from different cities)

SAN FRANCISCO BAY REGION, CALIF

Regular-price gasoline (continued)

I.B.P.	Distillation range, °F.						End point	Percent recovered	Item
	Percent recovered								
	10	20	30	50	70	90			
92	132	167	-	256	-	360	400	97	A Major ^{1/}
90	135	168	-	246	-	360	403	97	B do.
94	143	176	-	256	-	362	398	96.5	C do.
93	135	166	-	254	-	362	395	97	D do.
97	134	162	-	244	-	350	393	97	E do.
98	134	162	-	244	-	350	386	97	F do.
93	145	173	-	250	-	356	402	97.5	G do.
91	130	154	-	225	-	347	400	97	H do.
93	141	165	-	241	-	355	398	97.5	I Minor
98	142	174	-	249	-	373	420	98	J(1)do.
94	138	184	-	284	-	389	420	96	(2)do.
94	164	195	-	284	-	383	421	97.5	(3)do.
90	147	187	-	274	-	377	406	97	(4)do.
94	148	185	-	273	-	381	417	97.1	J(ave.)do.
90	130	160	-	240	-	350	400	97	K(1)do.
92	140	175	-	256	-	364	401	96.5	(2)do.
91	135	168	-	248	-	357	401	96.8	K(ave.)do.
90	134	162	-	247	-	346	388	98	L(1)do.
93	131	158	-	232	-	353	406	97	(2)do.
92	133	160	-	240	-	350	397	97.5	L(ave.)do.
93	137	167	-	248	-	358	399	97.1	Average

Premium-price gasoline (continued)

105	150	170	-	214	-	294	350	97.5	A Major ^{1/}
97	136	169	-	245	-	355	398	98	B do.
101	141	161	-	216	-	300	358	98	C do.
103	145	166	-	214	-	317	392	98	D do.
96	132	151	-	219	-	308	372	98	E do.
98	132	152	-	220	-	320	367	97.5	F do.
89	136	169	-	245	-	355	395	97	G do.
95	144	170	-	229	-	339	405	97.5	H do.
94	138	167	-	236	-	342	392	97.5	I Minor
98	140	162	-	215	-	296	356	97	J do.
99	138	158	-	226	-	320	371	97.5	K do.
101	150	178	-	240	-	338	377	97	L do.
98	140	164	-	227	-	324	378	97.5	Average

^{1/} Manufacturer.

TABLE 1.- Motor-gasoline survey, winter, 1936-37 - Continued
 (Data for samples from different cities)

SAN FRANCISCO BAY REGION, CALIF.

Third-grade gasoline

Item	Gravity		Sulphur, %	Reid vapor pressure, lb.	Copper- strip test	ASTM octane number
	Specific	°API				
A	0.740	56.9	0.190	9.1	Neg.	61.0
B	.756	55.6	.090	6.6	do.	60.5
C	.754	56.2	.120	8.3	do.	60.5
D	.751	56.9	.130	8.1	do.	62.5
E	.744	58.7	.077	7.3	do.	62.5
F	.746	58.3	.071	8.4	do.	62.0
G	.754	56.1	.250	7.4	do.	64.0
H	.738	60.2	.070	9.7	do.	62.5
I	.761	55.4	.091	6.3	do.	59.0
J(1)	.756	55.8	.220	7.0	do.	61.5
(2)	.753	56.4	.250	7.5	do.	64.0
(3)	.742	59.2	.064	9.5	do.	64.0
J(ave.)	.750	57.1	.178	8.0	do.	63.1
K(1)	.744	58.7	.092	7.5	do.	63.0
(2)	.753	56.4	.079	7.5	do.	62.0
K(ave.)	.748	57.6	.086	7.5	do.	62.5
L(1)	.748	57.6	.110	9.0	do.	62.0
(2)	.736	60.7	.051	9.6	do.	64.0
(3)	.738	60.1	.062	9.6	do.	63.5
(4)	.739	59.9	.029	8.4	do.	60.0
L(ave.)	.740	59.6	.063	9.2	do.	62.4
Average	.749	57.4	.118	8.0	do.	61.9

TABLE 1.- Motor-gasoline survey, winter 1936-37 - Continued
(Data for samples from different cities)

SAN FRANCISCO BAY REGION, CALIF.

Third-grade gasoline (continued)

I.B.P.	Distillation range, °F.						End point	Percent recovered	Item
	Percent recovered								
	10	20	30	50	70	90			
88	136	177	-	283	-	386	419	97	A Major ^{1/}
101	163	200	-	280	-	376	416	98	B do.
94	147	191	-	269	-	384	420	97	C do.
90	152	198	-	280	-	384	416	96	D do.
98	147	180	-	258	-	372	420	97.5	E do.
98	144	178	-	257	-	378	421	97	F do.
94	159	193	-	270	-	379	429	97	G do.
90	133	162	-	240	-	371	426	97	H do.
102	167	206	-	279	-	372	410	97.5	I Minor
100	160	200	-	275	-	376	424	98	J(1)do.
98	155	193	-	267	-	376	429	98	(2)do.
90	135	169	-	246	-	361	391	97	(3)do.
96	150	187	-	263	-	371	415	98	J (ave.)do.
99	147	179	-	257	-	373	426	97.5	K(1)do.
100	155	193	-	270	-	381	419	97	(2)do.
100	151	186	-	264	-	377	423	97.3	K (ave.)do.
88	142	185	-	270	-	374	409	96.5	L(1)do.
92	136	160	-	236	-	368	420	97	(2)do.
91	131	160	-	238	-	371	431	97	(3)do.
92	146	176	-	250	-	359	398	97	(4)do.
91	139	170	-	249	-	368	415	96.9	L (ave.)do.
95	135	186	-	266	-	377	419	97.2	Average

^{1/} Manufacturer.

TABLE 2. - Motor-gasoline survey, winter, 1936-37
(Average values for samples from different cities)

Regular-price gasoline

City	Gravity		Sulphur, %	Reid vapor pressure, lb.	Copper- strip test	ASTM octane number
	Specific	°API				
Boston	0.738	60.3	0.071	9.9	Neg.	70.2
New York734	61.4	.063	10.4	do.	70.6
Philadelphia .	.734	61.2	.051	10.0	do.	70.8
Baltimore732	61.7	.052	10.3	do.	70.4
Pittsburgh733	61.6	.065	10.0	do.	70.6
Cleveland731	62.2	.057	9.9	do.	70.1
Detroit726	63.5	.062	10.7	do.	69.8
Atlanta736	60.7	.060	8.7	do.	69.0
Chicago721	64.7	.047	11.2	do.	69.9
New Orleans ..	.734	61.2	.048	10.3	do.	70.3
St. Louis735	61.1	.058	10.9	do.	70.0
Minneapolis ..	.723	64.1	.062	11.4	do.	70.1
Omaha723	64.1	.038	10.7	do.	69.9
Tulsa725	63.8	.034	10.3	do.	69.9
North Texas ..	.734	61.4	.050	9.1	do.	69.2
South Texas ..	.737	60.6	.056	8.7	do.	69.2
Denver727	63.2	.035	10.4	do.	69.8
Salt Lake City	.733	61.4	.046	10.6	do.	70.0
Seattle737	60.5	.130	9.7	do.	71.4
Los Angeles ..	.743	58.9	.122	-	do.	70.5
San Francisco .	.743	58.9	.151	9.1	do.	70.5
Average732	61.7	.065	10.1	do.	70.1

TABLE 2. - Motor-gasoline survey, winter, 1936-37 - Continued
(Average values for samples from different cities)

Regular-price gasoline (continued)

I.B.P.	Distillation range, °F.							Percent		City
	Percent recovered						End point			
	10	20	30	50	70	90		Resid.	Loss	
92	137	164	191	239	283	349	398	1.1	1.9	Boston
89	133	163	192	255	288	349	396	1.1	1.9	New York
89	131	161	192	241	285	344	394	1.3	1.7	Philadelphia
88	129	159	188	238	284	348	397	1.1	2.2	Baltimore
89	136	164	190	234	279	341	387	1.0	1.4	Pittsburgh
89	123	161	189	237	284	351	404	1.1	1.8	Cleveland
87	125	150	177	229	280	350	396	1.0	2.0	Detroit
92	138	168	196	243	288	344	393	1.1	1.3	Atlanta
88	121	146	173	226	279	353	399	1.0	2.6	Chicago
91	134	164	194	247	293	355	398	.9	1.9	New Orleans
89	125	150	177	229	284	357	401	.9	2.6	St. Louis
86	120	146	172	226	283	356	401	.9	2.7	Minneapolis
87	122	149	177	232	282	328	400	.9	2.4	Omaha
89	126	152	173	232	285	356	402	1.1	2.4	Tulsa
92	138	169	197	247	293	354	400	1.0	2.1	North Texas
92	139	165	197	245	288	347	395	.9	1.7	South Texas
88	128	157	183	235	288	360	402	.9	2.6	Denver
78	116	146	178	234	289	364	434	1.0	3.8	Salt Lake City
86	127	157	188	242	291	357	400	1.0	2.8	Seattle
91	139	171	203	252	296	358	400	.8	—	Los Angeles
93	137	167	—	248	—	358	399	—	—	San Francisco
89	130	159	186	239	286	352	400	1.0	2.2	Average

TABLE 3. - Motor-gasoline survey, winter, 1936-37
(Average values for samples from different cities)

Premium-price gasoline

City	Gravity		Sulphur, %	Reid vapor pressure, lb.	Copper- strip test	ASTM octane number
	Specific	°API				
Boston	0.736	60.9	0.064	9.4	Neg.	77.7
New York734	61.2	.062	9.3	do.	77.8
Philadelphia .	.732	61.7	.034	9.2	do.	77.6
Baltimore745	58.5	.098	9.2	do.	77.0
Pittsburgh735	61.0	.069	9.3	do.	77.9
Cleveland728	62.9	.054	10.0	do.	76.5
Detroit727	63.0	.049	9.1	do.	76.1
Atlanta732	61.7	.028	8.1	do.	77.3
Chicago718	65.5	.043	10.5	do.	76.3
New Orleans ..	.733	61.6	.051	8.2	do.	78.5
St. Louis724	64.0	.061	9.9	do.	75.9
Minneapolis ..	.711	67.4	-	10.3	do.	77.9
Omaha719	65.2	.031	9.4	do.	77.0
Tulsa729	62.7	.040	8.8	do.	75.9
North Texas ..	.734	61.3	.037	8.5	do.	76.1
South Texas ..	.734	61.3	.043	7.7	do.	77.2
Denver725	63.6	.027	9.0	do.	76.4
Salt Lake City	-	-	-	-	-	-
Seattle734	61.3	.040	8.5	Neg.	80.5
Los Angeles ..	.737	60.5	.047	-	do.	80.0
San Francisco .	.734	61.3	.044	8.2	do.	80.0
Average730	62.3	.049	9.1	do.	77.5

TABLE 3. - Motor-gasoline survey, winter, 1936-37 - Continued
(Average values for samples from different cities)

Premium-price gasoline (continued)										
I.B.P.	Distillation range, °F.						End point	Percent		City
	Percent recovered							Resid.	Loss	
	10	20	30	50	70	90				
95	142	166	190	229	270	333	390	1.1	1.4	Boston
94	139	166	189	232	236	336	389	1.2	1.8	New York
88	131	159	186	230	271	332	388	1.0	1.3	Philadelphia
98	142	165	184	219	261	331	388	1.1	1.1	Baltimore
95	141	166	188	228	268	337	388	1.2	1.0	Pittsburgh
90	130	155	180	220	259	326	387	1.1	1.7	Cleveland
87	125	150	176	228	281	347	393	1.0	1.5	Detroit
92	135	164	189	236	275	329	390	1.0	1.4	Atlanta
90	122	144	168	214	263	335	391	1.0	1.8	Chicago
99	142	170	193	231	269	325	384	1.0	1.4	New Orleans
95	129	152	175	221	266	344	393	1.1	2.4	St. Louis
89	121	140	160	200	244	308	369	1.0	2.0	Minneapolis
91	127	150	173	218	264	331	384	.9	1.6	Omaha
95	131	156	181	229	282	355	407	1.0	2.0	Tulsa
90	133	165	194	242	285	342	394	1.0	1.2	North Texas
96	142	167	192	233	273	329	382	1.0	1.2	South Texas
94	131	155	179	227	278	345	398	.9	1.6	Denver
-	-	-	-	-	-	-	-	-	-	Salt Lake City
95	136	161	184	226	267	324	381	1.0	1.4	Seattle
97	143	169	194	234	272	327	380	.8	-	Los Angeles
98	140	164	-	227	-	324	378	-	-	San Francisco
93	134	159	183	226	268	333	388	1.0	1.5	Average

TABLE 4. - Motor-gasoline survey, winter 1936-37
(Average values for samples from different cities)

Third-grade gasoline						
City	Gravity		Sulphur, %	Reid vapor pressure, lb.	Copper- strip test	ASTM octane number
	Specific	API				
Boston	0.738	60.1	-	9.1	Neg.	63.0
New York	-	-	-	-	-	-
Philadelphia .	.736	60.9	0.042	8.5	Neg.	61.7
Baltimore	-	-	-	-	-	-
Pittsburgh ...	-	-	-	-	-	-
Cleveland731	62.0	.038	6.6	Neg.	54.1
Detroit734	61.3	-	6.9	do.	53.1
Atlanta738	60.3	.051	8.8	do.	62.9
Chicago734	61.4	.039	8.0	do.	54.7
New Orleans...	.736	60.9	.038	6.8	do.	60.0
St. Louis736	60.9	.028	8.0	do.	54.8
Minneapolis ..	.736	60.9	-	8.3	do.	53.5
Omaha731	62.0	.039	7.3	do.	52.4
Tulsa731	62.0	.022	7.8	do.	53.9
North Texas ..	.735	61.1	.037	7.6	do.	55.2
South Texas ..	.737	60.4	.042	6.7	do.	55.8
Denver736	60.7	.026	8.5	do.	54.4
Salt Lake City	.747	57.9	.038	8.4	do.	58.2
Seattle747	57.9	.120	8.7	do.	62.9
Los Angeles ..	.750	57.1	.102	-	do.	62.2
San Francisco .	.749	57.4	.118	8.0	do.	61.9
Average738	60.3	.046	7.9	do.	57.5

TABLE 4. - Motor-gasoline survey, winter 1936-37 - Continued
(Average values for samples from different cities)

Third-grade gasoline (continued)										
I.B.P.	Distillation range, °F.							Percent		City
	Percent recovered						End point			
	10	20	30	50	70	90		Resid.	Loss	
92	140	170	197	243	287	349	399	1.0	2.4	Boston
-	-	-	-	-	-	-	-	-	-	New York
91	137	166	194	239	284	346	398	1.0	1.4	Philadelphia
-	-	-	-	-	-	-	-	-	-	Baltimore
-	-	-	-	-	-	-	-	-	-	Pittsburgh
99	152	182	208	248	288	346	410	1.1	1.4	Cleveland
100	155	184	209	251	296	350	405	1.0	.9	Detroit
97	147	178	206	252	295	352	398	1.0	1.6	Atlanta
100	146	178	206	255	304	367	411	1.0	1.6	Chicago
97	148	178	204	247	292	354	404	1.0	1.3	New Orleans
99	130	196	205	255	306	370	413	.9	1.8	St. Louis
94	144	178	206	256	307	354	419	1.0	1.9	Minneapolis
97	141	168	196	245	295	348	414	.9	1.9	Omaha
95	140	170	196	245	297	365	416	1.0	1.8	Tulsa
94	147	182	211	252	290	346	403	1.0	1.4	North Texas
95	148	180	210	252	294	349	402	1.0	1.0	South Texas
92	143	179	208	256	306	369	410	1.0	2.4	Denver
82	126	163	196	257	319	380	412	1.0	2.2	Salt Lake City
91	141	178	213	265	314	379	421	1.0	2.3	Seattle
94	148	188	223	272	318	380	417	.8	-	Los Angeles
95	135	186	-	266	-	377	419	-	-	San Francisco
95	143	178	205	253	300	360	410	1.0	1.7	Average

TABLE 7. - Motor-gasoline survey, winter, 1935-36
(Summary of data of the survey)

Test	Regular-price gasolines			Premium-price gasolines			Third-grade gasolines		
	Min.	Range of 90%	Max.	Min.	Range of 90%	Max.	Min.	Range of 90%	Max.
Degrees A.P.I.	54.0	58.0 - 66.5	68.6	50.1	57.5 - 70.0	75.4	55.4	56.5 - 64.0	69.5
Sulphur content, %.	.014	.02- .15	.30	.006	.02- .08	.179	.016	.02- .20	.23
Reid vapor pressure, lb.	5.8	7.1 - 12.4	13.15	3.1	5.0 - 11.5	12.55	4.2	5.5 - 10.4	13.4
Copper-strip test .	-	Neg.	-	-	Neg.	-	-	Neg.	-
ASTM octane number.	61	68 - 71	72	69.1	73.7 - 80.5	81	33.5	50 - 66	69.1
Initial boiling point	79	82 - 98	100	76	84 - 107	128	78	84 - 104	128
10% recovered	108	114 - 144	155	110	115 - 150	178	113	127 - 161	179
20% recovered	124	132 - 175	186	117	134 - 176	201	128	150 - 196	205
30% recovered	141	156 - 206	218	124	147 - 209	224	146	183 - 224	233
50% recovered	178	207 - 255	266	143	180 - 252	261	184	230 - 276	282
70% recovered	192	267 - 308	316	181	225 - 301	314	223	270 - 326	337
90% recovered	245	335 - 377	388	264	273 - 359	378	292	330 - 393	399
End point	332	385 - 415	464	331	343 - 415	439	341	388 - 430	511
Dist. loss9	1 - 3.5	5.0	.5	1 - 3	3.6	.5	1 - 3.5	5.5

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UNITED STATES BUREAU OF MINES
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REPORT OF INVESTIGATIONS

RECOVERY OF POTASH FROM TAILING OF A PORPHYRY
COPPER PROPERTY



BY

G. L. OLDRIGHT, VIRGIL MILLER, HORST SIEG,
W. E. KECK, AND F. K. SHELTON

REPORT OF INVESTIGATIONS

DEPARTMENT OF THE INTERIOR - BUREAU OF MINES

RECOVERY OF POTASH FROM TAILING OF A PORPHYRY
COPPER PROPERTY^{1/}

By G. L. Oldright,^{2/} Virgil Miller,^{3/} Horst Sieg,^{4/}
W. E. Keck,^{5/} and F. K. Shelton^{6/}

Potash is one of the principal plant foods and is necessary for the growth of most vegetables, grains, and fruits. For many years, this country depended entirely upon foreign sources for its supply of potassium compounds. However, during the last 15 years the domestic production of potash has gradually increased, as shown in table 1.^{7/} Imports now supply only half of the American market. In 1936, the equivalent of 207,000 tons of K_2O was imported, and 210,000 tons was produced domestically. In 1935, domestic consumption was approximately 420,000 tons of potash, a 57 percent increase over that for 1934 and surpassing previous peaks of 1928 and 1930.

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- 1/ The Bureau of Mines will welcome reprinting of this paper provided the following footnote acknowledgment is used: "Reprinted from U. S. Bureau of Mines Report of Investigations 3349."
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 - 7/ Hedges, J. H., Potash: Minerals Yearbook 1935, Bureau of Mines, p. 1148.

TABLE 1. - Domestic production of potash,
in short tons

Year	Crude salts	K ₂ O content
1922	25,176	11,714
1923	39,029	20,215
1924	43,734	22,903
1925	51,565	25,448
1926	46,324	23,366
1927	76,819	43,510
1928	104,129	59,910
1929	107,820	61,590
1930	105,810	61,270
1931	133,920	63,880
1932	143,120	61,990
1933	333,110	143,378
1934	275,732	144,342
1935	357,974	192,793
1936 (estimate)	400,000	210,000

The three main domestic producers are the American Potash and Chemical Corporation at Searles Lake, Calif., the U. S. Industrial Chemical Co. at Baltimore, Md., and the United States Potash Co. and the Potash Co. of America at Carlsbad, N. Mex. The first produces potash from lake brines, the second from distillery wastes, and the third and fourth from bedded deposits. A large tonnage of potash minerals has been developed in the natural bedded mineral deposits of eastern New Mexico and western Texas. The market in which a given producer of potash can compete successfully will depend upon relative costs, that for freight being a large item.^{7A/}

The manufacturer of potash salts, who can also market mixed or balanced fertilizers containing any of the plant foods for which there is a large demand, will have many advantages. He not only can sell more potash, but by selling a complete plant food he greatly increases the opportunity of introducing his potash into new markets in particular. It is noteworthy that the source of potash in Utah, on which the following experimental work was done, is near large supplies of such raw materials for the manufacture of a complete fertilizer as coal, natural gas, phosphate rock, sulphur dioxide and by-product ammonia.

The soil of Utah and other Western States has not been depleted of potash to as great an extent as in the older regions of the country. However, even in Utah the most intensely cultivated soil is being exhausted. Some of the land is producing less than one-half of its former yields, and consequently the cost of production from these areas has increased greatly.

^{7A/} Bourquin, J. J. The Potash Industry in the United States. Min. Cong. Jour., vol 23, Jan. 1937, pp. 57-59. (Events divided into five distinct epochs.)

Sugar beets are one of the most important crops of the West. Their production has been curtailed to such an extent by the exhaustion of soil fertility that several sugar factories have moved from various districts. Potash is essential for the growth and development of this crop, and a cheap source of this fertilizer ingredient is desirable. Therefore, it is of importance to States, such as Utah, that are distant from sources of production to look into their own potash resources.

The alunite deposits have been investigated.^{7B/} Among other resources are certain of the feldspars (orthoclase and microcline) and potash mica (muscovite). These minerals contain from 11 to 16 percent potash combined with alumina and silica and cannot be used directly as fertilizers, as they are insoluble in water. (A small amount of alunite is used directly each year as a fertilizer.)

In many instances, these potash minerals occur as gangue in ores, such as those of porphyry copper. Therefore, the mill tailings from many copper concentrators may be considered as possible sources of potash. Enormous tonnages of such tailings, containing 5 to 10 percent K_2O , are available. For example, when operating at full capacity the two concentrators of the Utah Copper Co. produce 75,000 tons daily. Such material can be handled cheaply, and, in addition, there is no charge for mining and crushing. Therefore, the Bureau of Mines, in cooperation with the University of Utah, some years ago investigated the feasibility of recovering potash from such mill tailings.

The tailings used in these experiments were obtained from the Utah Copper Co. They were selected because they contained approximately 6 percent potash, were finely ground, were piled so as to be easily available, and were representative of the product discarded from many mills treating porphyritic ores. A typical analysis is given in table 2.

TABLE 2. - Analysis of tailings from Utah Copper Co.

Constituent	Percent
Cu	0.165
SiO_2	72.20
Al_2O_3	16.00
Fe	0.95
S	0.55
Na_2O	1.10 ^{1/}
K_2O	5.90 ^{1/}
Total	98.835

^{1/} The tailings used in this research contained 6.5 per cent K_2O .

^{7B/} Knickerbocker, R. G., and Koster, J. Electrometallurgical Studies on the Treatment of Alunite: Rept. of Investigations 3322, Bureau of Mines, 1936, pp. 39-64.

RECOMMENDED PROCEDURE

After a laboratory investigation of various methods for the recovery of potash from the tailings studied, the following procedure was recommended as being practical and offering possibilities of successful development on a larger scale:

1. Heat the tailing from 900° to 950° C. in a rotary kiln.
2. Melt sodium chloride in a rotary kiln or other direct-fired equipment.
3. Place a mixture of 65 percent hot ore 32 percent sodium chloride, and 3 percent coal in a well-insulated soaking pit.
4. Feed additional mixture to the top and remove it from the bottom of the pit continuously at such a rate that passage through the pit takes 3 to 6 hours. The temperature at the bottom of the pit should not be under 800° C.
5. Cool the calcine to 400° or 500° C. in a closed container (to prevent volatilization of chlorides), followed by stirring and further cooling to 200° to 250° C. in air (to permit oxidation of objectionable chlorides).
6. Leach the calcine countercurrently with water near the boiling point so that a saturated solution results.
7. Cool the leach liquor to precipitate most of the potassium chloride.
8. The mother liquor can be recirculated or evaporated and the resulting mixtures of sodium chloride with some potassium chloride used as the chloridizing agent.
9. The potassium chloride content of the precipitate from the mother liquors may be enriched to over 90 percent KCl by washing with cold water.

EFFECT OF POSSIBLE IMPURITIES ON
PLANT GROWTH

Impurities likely to occur with potassium chloride made by such a process are sodium chloride, calcium chloride (if calcium compounds are used in roasting), iron and aluminum compounds, and negligible quantities of other substances.

Rusche^{8/} found that calcium chloride hindered the germination of beet seed, but no more than potassium chloride affected germination of alfalfa and clover seeds. Five hundred pounds of sodium chloride per acre increased the yield and quality of sugar beets.^{2/}

Miyake^{10/} found that sodium chloride and potassium chloride had a toxic effect on the growth of rice plants if present separately in concentrations greater than

^{8/} Rusche, A., Jour. Landw., Vol. 60, no. 4, 1912, pp. 305-365.

^{2/} Brock, A.J., Salt Treatment for Control of Diseases of Sugar Beet Seedlings: Facts about Sugar, vol. 25, 1930, pp. 705-706.

^{10/} Miyake, K., The Influence of Salts Common in Alkali Soils upon the Growth of the Rice Plant: Jour. Bio. Chem., vol. 16, no. 2, 1913, pp. 235-263.

1/100 normal. Calcium chloride had a beneficial effect in concentrations of 1/1000 to 1/5000 normal, and sodium chloride gave similar results from 1/50 to 1/100 normal. If potassium chloride were added to either calcium chloride or sodium chloride, the toxic effect of the calcium or sodium ions was overcome and plant growth was affected less.

Aluminum salts must be removed, as they have been found to be markedly toxic to the growth of barley,^{11/} sugar cane,^{12/} and wheat.^{13/} Ferrous chloride^{14/} has a marked beneficial effect when its concentration in the soil is less than 0.1 percent iron; it has a toxic effect on the function of plants if this concentration is exceeded. Ferric chloride is decidedly toxic, and a concentration in the soil of 0.2 percent iron will prevent entirely the growth of plants. Other iron compounds have little or no effect on plant growth.

Therefore, aluminum salts and iron chlorides must be removed completely from the leach liquors or made insoluble at the start. It is desirable to remove sodium chloride and calcium chloride from the precipitated KCl to reuse them in the process and to save freight on inert material.

EXPERIMENTAL DATA

Chloridizing Roast

Many methods have been proposed for rendering soluble the potash of orthoclase, mica, leucite, and other potash-bearing minerals. These methods include roasting with sodium chloride or calcium chloride.^{15/} Usually calcium carbonate or calcium oxide is added to the mixture.

The applicability of these procedures to the present problem was studied thoroughly. One gram of minus 200-mesh ore was mixed with the selected reagents, placed in a tightly covered platinum (J. Lawrence Smith) crucible, and heated. The resulting calcine was leached with 100 cm³ of boiling water for one-half hour and filtered. The residue was washed with water, and the washings were combined with the filtrate.

On using calcium chloride and roasting for 1 hour at 800° C., up to 88 percent of the potash was extracted, as shown in table 3. Addition of calcium carbonate to the mixture increased extraction especially with the lower concentrations of calcium chloride. Potash recovery was increased by calcium chloride in solution rather than dry.

^{11/} Conner, S.D., and Sears, O.H., Aluminum Salts and Acids at Varying Hydrogen-ion Concentrations in Relation to Plant Growth in Water Culture: Soil Sci., vol. 13, no. 1, 1922, pp. 23-24.

^{12/} McGeorge, W.J., The Influence of Aluminum, Iron, and Manganese Salts upon the Growth of Sugar Cane, and Their Relation to Infertility of Acid Island Soils: Bull. Hawaii Sugar Planters Assoc. vol. 49, 1925, p. 95.

^{13/} Barnett, R.M., The Influence of Soluble Aluminum Salts on the Growth of Wheat Seedlings in Shives R₂C₂ Solution: New Jersey Sta. Rept., 1923, pp. 255-258.

^{14/} Voelcker, J.A., Pot Culture Experiments 1918: Royal Agr. Soc. England, Woburn Exp. Sta. vol. 78, 1918, pp. 17-24.

^{15/} Cushman, A.S., and Coggeshall, G.W., The Production of Available Potash from Natural Silicates: Jour. Ind. Eng. Chem., vol. 4, no. 11, 1912, pp. 821-827.

TABLE 3. - Recovery of K_2O from tailing after 1 hour
chloridizing roast with $CaCl_2$ and $CaCO_3$ at 800°

$CaCl_2$ added	percent of tailing	K_2O recovery, percent		
Dry	In solution	$CaCO_3$, none	$CaCO_3$, 25% of tailing	$CaCO_3$, 50% of tailing
10	---	5.0	14.3	25.6
15	---	---	37.5	32.8
---	25	---	55.0	---
25	---	32.6	53.9	56.9
50	---	61.5	79.0	81.9
---	75	---	93.5	---
75	---	85.0	87.9	---
100	---	88.3	---	---

On using sodium chloride, a much cheaper reagent available in large quantities near the tailing dumps, much less potash was extracted, as shown in table 4. As with calcium chloride, calcium carbonate increased potash recovery. Increasing the temperature from 800° to 900° had the same effect. At the lower temperature, more potash was recovered, as the time of roasting was increased when no calcium carbonate was used; at the higher temperature, after equal portions of ore and salt were roasted with varying amounts of limestone, potash solubility decreased with increased time of roasting. However, the best percentage recovered was only 65 or 70 percent.

TABLE 4. - Recovery of K_2O from tailing after chloridizing roasts with
NaCl and limestone.

NaCl added, percent of tail- ing	K ₂ O recovery, percent											
	CaCO ₃ , none		CaCO ₃ , 10% of tailing		CaCO ₃ , 25% of tailing		CaCO ₃ , 50% of tailing		CaCO ₃ , 75% of tailing		100% of tailing	
	800° C.		800° C.		800° C.		800° C.		800° C.		800° C.	
	1 hr.	2 hrs.	1 hr.	2 hrs.	1 hr.	2 hrs.	1 hr.	2 hrs.	1 hr.	2 hrs.	1 hr.	2 hrs.
10	10.0	---	---	---	---	---	---	---	---	---	---	---
25	23.8	31.6	39.7	21.4	44.3	41.3	41.3	---	45.3	41.3	---	---
50	33.5	45.1	49.5	45.6	53.7	52.6	52.6	---	49.4	51.7	---	---
75	43.4	50.3	56.6	54.4	---	---	---	---	---	---	---	---
100	46.8	53.9	60.3	58.0	48.2	68.6	54.9	41.1	51.4	59.4	65.8	57.0

The use of lime in place of calcium carbonate did not change potash recovery appreciably. Substantially more potash was recovered after roasting with mixtures of sodium chloride and calcium chloride than when either reagent was used alone, as shown in table 5. Addition of calcium carbonate gave still further increases and at the same time permitted the use of smaller amounts of calcium chloride. On increasing the temperature to 900° and reducing the time of roasting to one-half hour, 98.4 percent of the potash was extracted.

TABLE 5. - Recovery of K₂O from tailing after chloridizing roast

NaCl added, percent of tailing	CaCl ₂ added, percent of tailing	CaCO ₃ added, percent of tailing	K ₂ O recovery, percent	
			800°, 1 hour	900°, 0.5 hour
0	10	0	5.0	---
0	25	0	32.6	---
0	50	0	61.5	---
0	100	0	88.3	---
10	25	25	79.3	---
20	25	25	82.0	---
25	10	25	54.6	98.4
30	25	25	83.5	---
40	25	25	80.6	---
50	0	0	33.5	---
50	10	0	45.3	---
50	20	25	80.2	---
50	25	0	65.9	---
50	25	25	80.2	---
50	50	0	74.1	---
100	0	0	46.8	---

When the charge was moist, less potash and chlorine were recovered: Therefore, it was found desirable to dry the ore before treatment (see table 6)

TABLE 6. - Recovery of K₂O from tailing after 1-hour chloridizing roast at 825-850°C.; charge-ore, 55 percent, CaCl₂, 14 percent; CaCO₃, 14 percent; and NaCl 16 percent.

	Dried at	
	Room temp.	125° C.
K ₂ O recovery..... percent.....	81.0	86.4
Cl recovery..... do.....	62.0	87.5

Serious objections to the use of calcium chloride may be cited. Unless the reagent may be regenerated cheaply in the process, it is more expensive than common salt. Also, calcium chloride forms complex salts from which it is difficult to crystallize pure KCl , a step that is simple when sodium and potassium chlorides are the main salts present.

When the tailings containing potassium silicates were heated in an enclosed vessel with small amounts of coal and common salt alone, most of the potassium was converted to the chloride. The most favorable proportions were tailings, 65 percent; common salt, 32 percent; and coal, 3 percent (see table 7).

TABLE 7. -- Recovery of K_2O from tailing after chloridizing roast with $NaCl$ and coal at 800° - 810° C.

NaCl added, percent of tailing	Coal added, percent of tailing	Recovery, percent											
		K_2O						Cl_2					
		1 hr.	2 hrs.	3 hrs.	4 hrs.	5 hrs.	6 hrs.	1 hr.	2 hrs.	3 hrs.	4 hrs.	5 hrs.	6 hrs.
20	20	--	--	62.0	--	--	--	--	--	97.2	--	--	--
30	2.5	58.1	63.5	--	67.5	68.5	77.0	96.7	95.3	--	93.4	92.5	90.3
30	20	--	--	70.0	--	--	--	--	--	95.3	--	--	--
40	20	--	--	77.0	--	--	--	--	--	95.8	--	--	--
50	0	--	--	55.0	--	--	--	--	--	93.7	--	--	--
50	2.5	--	--	90.0	--	--	--	--	--	93.2	--	--	--
50	5	--	--	88.2	--	--	--	--	--	97.5	--	--	--
50	10	--	--	85.6	--	--	--	--	--	98.9	--	--	--
50	20	--	--	83.3	--	--	--	--	--	98.3	--	--	--
50	40	--	--	59.3	--	--	--	--	--	99.2	--	--	--
50	80	--	--	51.0	--	--	--	--	--	99.5	--	--	--
60	20	--	--	87.5	--	--	--	--	v	98.0	--	--	--

In the foregoing experiments, the charge was heated indirectly to simulate retort roasting. Further experiments were made, in which a mixture of 65 percent ore, 32 percent sodium chloride, and 3 percent coal was roasted with natural gas in rotary kilns for 2, 3, 4, and 5 hours at 800° C. One kiln was 4 inches in inside diameter and 2 feet long; another was 1 by 4 feet. The depth of the charge in the small kiln was $1\frac{1}{2}$ inches and in the large one 6 inches. Less than 20 percent of the potash was recovered in each case, and losses of the chloridizer were great, due to the large volume of gases passing through the kiln.

To avoid these losses, the possibility of heating ore and chloridizer separately, then mixing and allowing them to react in a quiet atmosphere in a soaking pit was investigated. Maximum temperatures must be set to avoid clinkering of the ore and volatilization of the chloride. At 900° the ore began to cohere slightly, and at $1,000^{\circ}$ it clinkered badly. Published data^{16/} indicate that the losses by volatilization on heating sodium chloride to the melting point (about 800° C.) in a quiet atmosphere would be small. No difficulties would be expected in heating the ore to 900° and the salt to 800° before mixing them.

^{16/} Landolt, H.H., and Börnstein, R., *Physikalisch-Chemische Tabellen*, 5 Aufl., II, p. 1347; International Critical Tables, McGraw-Hill Book Co., New York, 1st ed., vol. 3, 1928, p. 349.

To 2,000 grams of ore heated to 900° was added 100 grams of coal and 1,000 grams of hot, crystalline sodium chloride. After mixing, the pit was covered and allowed to stand about 3 hours until the charge had cooled to 800° C. A similar experiment was made, in which molten sodium chloride was used. In the first experiment 41 percent of the K_2O was recovered and in the latter, 47.7 percent. Therefore, all subsequent experiments were made with molten sodium chloride.

Results of tests to determine the optimum amounts of coal and sodium chloride when treating in a soaking pit are shown in table 8. In no case was the potash recovery satisfactory, although much more was extracted than by roasting in a rotary kiln.

TABLE 8. -- Recovery of K_2O from tailings after
treatment in a soaking pit.

NaCl added, percent of tailing	Coal added, percent of tailing	Temp. of ore at addition of coal or NaCl, $^{\circ}$ C.	Time above 800° , hours	K_2O recovery, percent
30	5	890	3.8	30.2
30	5	900	3.5	36.8
40	5	895	3.6	48.0
40	5	925	4.0	45.9
50	3	945	4.4	36.1
50	5	890	4.1	47.2
50	5	900	3.8	47.7
50	10	905	3.8	40.1
50	10	935	4.5	38.8
50	15	925	3.25	39.9
50	15	930	3.6	36.4

1/ About.

On sampling successively lower layers of the charge, the percentage of K_2O recovered increased progressively almost to the bottom (see table 9). This indicates that the failure to recover potash was due to loss of the chloridizer. (This loss was checked by absorption tests.) The temperature of ore in the charge while in the soaking pit should be above 800° C.

TABLE 9. -- Recovery of K_2O from various levels in the soaking pit after holding at 800 to 900° , 2 to 4 hours

Place of sample	K_2O recovery percent		
	Charge 4 inches deep	Charge 8 inches deep	Charge 4 inches deep, covered with 1 inch of coal
Top.....	34.6	15.2	29.7
One-fourth distance from top to bottom...	69.6	57.4	75.0
One-half distance from top to bottom.....	74.2	78.9	77.9
Three-fourths distance from top to bottom	88.7	89.2	93.4
Bottom	83.8	87.0	79.4
Entire calcine.....	55.2	52.4	89.4

These tests indicate that in practice the loss of the chloridizer could be prevented if the charge were roasted in a deep column in the soaking pit discharged from the bottom.

Leaching of the Calcine

Using calcine prepared with calcium chloride, calcium carbonate, and sodium chloride, more potash was dissolved by dilute sodium hydroxide than by sulphuric acid, which, in turn, was superior to plain water (see table 10). However, the acidic leach dissolved large quantities of deleterious iron and aluminum, while the increased amount of potash dissolved by sodium hydroxide solutions was not worth the extra cost involved. Milk of lime gave better results than water but was inferior to sodium hydroxide.

TABLE 10. - Recovery of K₂O from tailing after 1 hour chloridizing roast at 300°, leaching with various reagents at boiling point; time of leach, 1/2 hour

NaCl added, percent of tailing	CaCl ₂ added, percent of tailing	CaCO ₃ added, percent of tailing	Recovery, percent of				
			K ₂ O			Chloride	
			2.7 cm ³ H ₂ SO ₄ per 100 cm ³ H ₂ O	H ₂ O	1.866 g NaOH per 100 cm ³ H ₂ O	H ₂ O	1.866 g NaOH per 100 cm ³ H ₂ O
10	25	25	85.0	79.0	82.9	46.7	47.9
20	25	25	88.4	82.0	90.4	44.6	46.9
30	25	25	87.3	83.6	88.6	44.5	46.8
40	25	25	87.6	80.6	91.6	45.4	46.8
50	25	25	86.3	80.2	96.2	44.7	45.6

For all subsequent leaching experiments, a calcine was prepared from a mixture of 1,000 grams ore, 250 grams calcium chloride, 250 grams calcium carbonate, and 300 grams sodium chloride, held at 800 to 900° for 1 hour, and then cooled to room temperature. Water was used as the leaching agent.

No difficulties were encountered in leaching. The solutions passed through the calcine freely, leaving a spongy residue.

The leaching solutions, unfortunately, contained iron and aluminum compounds, which are toxic to plant growth unless the calcine was roasted at 300 to 400° in an oxidizing atmosphere before it was leached. Ferrous chloride can be oxidized to ferric oxide,¹⁷ which is not dissolved by the leaching agent. Ferric

¹⁷ Mellor, J.W., Comprehensive Treatise of Inorganic and Theoretical Chemistry: Longmans, Green, London, vol. 5, 1924 ed., p. 316.

Friend, J.N., Text-book of Inorganic Chemistry: Chas. Griffin & Co., Ltd., London, vol. 9, 1922, part 2, p. 94.

Roscoe, H.E., and Schorlemmer, C., Treatise on Chemistry: Macmillan, London, 6th ed., vol. 2, 1923, p. 1276.

Landolt, H.H., and Bernstein, R., Physikalisch-Chemische Tabellen: Vol. 1, pp. 339, 343, and 349.

International Critical Tables, McGraw-Hill Co., New York, 1st ed., vol. 3, 1928, p. 349.

and aluminum chlorides can be volatilized at temperatures at which losses of the chlorides of potassium, calcium, and sodium are not excessive. After the experimental calcine was roasted at 300 to 400° for 1 hour in air and then leached, the effluent solutions contained no soluble iron or aluminum salts. The same results could be obtained in a commercial process by cooling the calcine from 400° to 200° in air.

The pregnant solutions were too dilute to precipitate potassium chloride on cooling to room temperature. Batch experiments to simulate counter current leaching were conducted by dividing the leach liquor into 20 cm³ portions and adding them to a fresh batch of calcine in the same order in which they had been taken from the preceding one. This leach cycle was repeated six times. Salts began to precipitate from the leach liquors after the third pass and were removed by decantation. Table 11 shows the additions of liquor during the experiment. All solutions were kept at about 100° C. The liquors were in contact with each batch about 2 minutes.

During the treatment, 19.9 grams of dry salts were precipitated, and on cooling to 25° C., 31 grams more were recovered.

TABLE 11. - Amount of liquor added and recovered during experiments simulating countercurrent extraction

Batch	Liquor added, cm ³	Liquor recovered, cm ³	Cl, in leach residue percent
1	550	440	.1/
2	410	340	0.3
3	340	280	.6
4	280	220	.8
5	220	180	7.0
6	180	140	14.1

1/ Trace.

These salts contained only about 35 percent potassium chloride. On recrystallizing, the product contained 44.8 percent KCl and 0.8 percent CaCl₂

The solubilities of potassium chloride and sodium chloride in the same solution are affected greatly by the presence of calcium chloride^{18/} (see table 12). This accounts for the small percentage of potassium chloride obtained by cooling the leach liquors from these experiments. Assuming (1) that 90 percent of the potash in the ore was converted to potassium chloride, (2) that 75 percent of the conversion was due to calcium chloride and 25 percent to sodium chloride, (3) that all chlorides were volatilized equally during roasting, and (4) that extraction was complete, the ratio of the equivalents of calcium chloride to potassium chloride to sodium chloride would be 5 to 1 to 6 in the hot leach liquor. On cooling to 25°, the ratios would be 292 to 22 to 2 (see table 12), provided there was enough calcium chloride to saturate the solution. The precipitated salt would contain

^{18/} International Critical Tables, McGraw-Hill Co., New York, vol. 4, 1928, p. 331.

little calcium chloride, much sodium chloride, and slightly less potassium chloride, which analysis showed to be the case.

TABLE 12. - The system KCl - CaCl₂ - NaCl - H₂O

Solid phases	Liquid phase - M per 100 H ₂ O			
	0.5 M CaCl ₂	M KCl	M NaCl	Temp., °C.
NaCl plus KCl.....	0	39	89	25
NaCl plus KCl plus CaCl ₂ ·2 H ₂ O..	498	22	1	83
NaCl plus KCl plus CaCl ₂ ·6 H ₂ O..	292	22	2	25

A second leaching test was made. At the conclusion, the precipitated salts were combined with the liquor, cooled to 70° C., and held for 3 hours. The precipitated salts were removed. The solution was cooled to 25° C. and held there for 12 hours. The resulting precipitate was removed. Both batches of salts were washed with cold water and analyzed.

Table 13 shows that the salts precipitated between 25° and 70° C. when washed with cold water were a commercial product.

TABLE 13. - Analyses of the products obtained by leaching
of the calcine from chloridizing roasting

	KCl	CaCl ₂ ^{1/}	Cl	NaCl ^{2/}
+70° C. salts..... percent	23.7	8.5	55.0	63.8
25° - 70° C. salts..... do.	61.9	2.2	50.7	33.2
Mother liquor..... grams per liter	45.0	569.0	391.0	8.3
Washed salts, + 70° C..... percent	45.3	1.1	54.3	52.6
Washed salts, 25° - 70° C..... do.	91.5	0.3	47.1	5.8
Wash solution from +70° C. salts.. grams per liter	90.0	119.0	204.0	141.9
Wash solution from 25° - 70° C., salts..... grams per liter	152.0	77.0	252.0	215.3

1/ Assuming that all of the calcium was present as calcium chloride.

2/ Computed from chlorine content.

Data in this paper show that the potash in the Utah Copper tailing may be converted to the chloride by roasting with common salt alone without any calcium chloride. The solubility relations shown in table 14^{19/} indicate that potassium chloride will deposit by simply cooling a saturated solution of potassium chloride and sodium chloride.

19/ Seidell, A. Solubilities of Inorganic and Organic Compounds: D. Van Nostrand Co., New York, 2nd ed., 1910, p. 521.

TABLE 14. - Equilibrium conditions in the system KCl - NaCl - H₂O

Temp., ° C.	Grams per 100 grams H ₂ O			
	KCl		NaCl	
	Precht & Wittgen, 1881	Estard, 1897	Precht & Wittgen, 1881	Estard, 1897
0	11.2	11.2	30	30
20	14.7	13.8	29.2	31
40	19.5	17.	28.2	31.9
60	24.6	20.6	27.2	32.8
90	32.9	28.4	26.1	32.3
100	34.7	32.3	25.8	30.6

Complete methods used for separating potassium chloride from comparable chloride liquors in plant practice are given in some of the references in the appended bibliography.

Other Methods for Recovering Potash from Tailings

Concentration. - Sedimentation tests showed no appreciable concentration of the potash in any size.

Flotation with varying amounts of sodium oleate effected no concentration of the potash. (The pyrite was removed first with 0.25 pound of sodium ethyl xanthate plus 0.25 pound of terpinol per ton of ore.) It should be noted, however, that only a few tests of a preliminary nature were made. The art of the flotation of nonmetallics is advancing so rapidly that the possibilities of the method should be explored much farther.

Leaching. - Three series of leaching tests were made, using a solution containing 50 percent sulphuric acid and recirculating for various periods up to 24 hours. Unsized ore was used in one series of tests and material through 200-mesh in the other two. In one of the latter, the ore was held at 850° to 900° C. for 1 to 3 hours before leaching.

Under the conditions selected, only 30 to 35 percent of the potash was dissolved by protracted treatment in strong sulphuric-acid solution. Size of material or preheating for 1 hour made little difference. After preheating longer, less potash dissolved.

Sulphating Roast. - Tailings were roasted with an excess of sulphuric acid, and the calcine was leached with water. Only 25 percent of the potash was recovered.

SUMMARY AND CONCLUSIONS

Immense quantities of tailings from porphyry copper properties, containing 5 to 10 percent K_2O , are available in the vicinity of Salt Lake City, Utah, and other mining districts of the West. They are finely ground and can be reclaimed for a few cents per ton.

Using tailings from the mill of the Utah Copper Co., a process was developed on a laboratory scale in which about 90 percent of the potash was recovered in a 90 percent KCl product. The other raw materials are salt and coal. These can be obtained readily at a low cost from near-by deposits.

The process, which is outlined in detail above, involves separate heating of tailings and salt, mixing with coal in a soaking pit (800°), cooling to 400° in a closed container, cooling to 200° in air, leaching, cooling of liquors to precipitate KCl, and washing and drying of crystals.

The technical feasibility of the process has been shown only on a laboratory scale. Before commercial possibilities can be evaluated, large-scale tests should be made.

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The first part of the paper is devoted to a general discussion of the problem of the origin of life. It is shown that the problem is one of the most important and most difficult in the history of science. The second part of the paper is devoted to a detailed discussion of the problem of the origin of life. It is shown that the problem is one of the most important and most difficult in the history of science. The third part of the paper is devoted to a detailed discussion of the problem of the origin of life. It is shown that the problem is one of the most important and most difficult in the history of science. The fourth part of the paper is devoted to a detailed discussion of the problem of the origin of life. It is shown that the problem is one of the most important and most difficult in the history of science. The fifth part of the paper is devoted to a detailed discussion of the problem of the origin of life. It is shown that the problem is one of the most important and most difficult in the history of science. The sixth part of the paper is devoted to a detailed discussion of the problem of the origin of life. It is shown that the problem is one of the most important and most difficult in the history of science. The seventh part of the paper is devoted to a detailed discussion of the problem of the origin of life. It is shown that the problem is one of the most important and most difficult in the history of science. The eighth part of the paper is devoted to a detailed discussion of the problem of the origin of life. It is shown that the problem is one of the most important and most difficult in the history of science. The ninth part of the paper is devoted to a detailed discussion of the problem of the origin of life. It is shown that the problem is one of the most important and most difficult in the history of science. The tenth part of the paper is devoted to a detailed discussion of the problem of the origin of life. It is shown that the problem is one of the most important and most difficult in the history of science.



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REPORT OF INVESTIGATIONS

DEPARTMENT OF THE INTERIOR - BUREAU OF MINES

PRODUCTION OF EXPLOSIVES IN THE UNITED STATES
DURING THE CALENDAR YEAR 1936^{1/2/}

By W. W. Adams^{3/} and V. E. Wrenn^{4/}

Production of explosives in the United States during the calendar year 1936, as represented by reports of sales by manufacturing companies, totaled 391,604,500 pounds, an increase of 27 percent over the amount produced during 1935. This quantity included 81,698,375 pounds of black blasting powder (granular and pellet), 262,047,106 pounds of high explosives other than permissibles, and 47,859,019 pounds of permissible^{5/} explosives.

The manufacturers' reports to the Bureau of Mines classify the sales of explosives according to five main classes of work in connection with which the explosives were used.

Coal mines used 87 percent of the total amount of black blasting powder sold, bituminous-coal mines 79 percent, and anthracite mines 8 percent; coal mines also consumed 98 percent of all permissible explosives and 9 percent of the high explosives other than permissibles. The coal-mining industry consumed 16 percent more explosives in 1936 than in the preceding year, the increase at bituminous mines being 21 percent and that at anthracite mines 4 percent.

Metal mines used 29 percent of the high explosives other than permissibles and less than 1 percent of the black blasting powder and permissible explosives. The total quantity of the three classes of explosives used in metal mining in 1936 was 36 percent more than the quantity so used in 1935.

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- ^{1/} The Bureau of Mines will welcome reprinting of this paper, provided the following footnote acknowledgment is used: "Reprinted from U. S. Bureau of Mines Report of Investigations 3350."
^{2/} In this publication "production" of explosives means the total sales during the year as reported by manufacturers.
^{3/} Supervising statistician, employment statistics section, U. S. Bureau of Mines.
^{4/} Employment statistics section, U. S. Bureau of Mines.
^{5/} "Permissible explosives" are high explosives that have passed certain tests prescribed by the Bureau of Mines and are therefore considered reasonably safe for use in gassy and dusty coal mines.

Quarries and nonmetallic-mineral mines consumed 19 percent of the high explosives other than permissibles, approximately 3 percent of the black blasting powder, and slightly over 1 percent of the permissibles sold. The total quantity of explosives used by these industries increased 45 percent when compared with 1935.

Most of the explosives used in railway and other construction work were high explosives other than permissibles. Reports from the manufacturers showed that nearly 38 percent of the total sales of this class of explosives were used in construction work. Such work also required more than 9 percent of the black blasting powder but considerably less than 1 percent of the permissible explosives. The total amount of explosives used in construction work in 1936 represented an increase of 27 percent over 1935.

Miscellaneous types of work used 5 percent of the high explosives other than permissibles and less than 1 percent of the black powder and of the permissibles. Compared with 1935, the total quantity of explosives used in 1936 represented an increase of 45 percent.

Sales of black blasting powder have declined almost steadily since 1917, the peak production year. Although the amount sold in 1936 was 18 percent more than was sold in 1935, it was 70 percent less than the total sales in the peak year, 1917.

Sales of high explosives other than permissibles have maintained a fairly level course since 1912, the first year for which figures are available, although a large volume of business was done from 1923 to 1930 and a serious drop in sales occurred in 1932. Maximum production was recorded in 1929, when sales reached 326,993,276 pounds, and the lowest point during the 25-year period was in 1932, when only 137,907,607 pounds were sold. Since 1932 sales have increased each year, except in 1935, reaching 262,047,106 pounds in 1936, 20 percent less than the quantity sold in the peak year of 1929.

Production of permissible explosives increased from 24,630,270 pounds in 1912, the first year of the 25-year period under consideration, to a maximum of 67,685,416 pounds in 1926. In the depression year, 1932, sales dropped to 32,225,193 pounds but rose to 47,859,019 pounds in 1936, an increase of 49 percent over 1932 and an increase of 22 percent over 1935.

Table 1 shows the percentage of increase or decrease in the quantities of explosives used for various purposes during 1936, compared with 1935.

Tables 2 to 5 show the amount and use, by States, of each of the three classes of explosives manufactured and consumed in the United States during the calendar year 1936.

TABLE 1. - Percentage increase or decrease in production of explosives in 1936 compared with 1935

Use	Block blasting powder	Permissible explosives	Other high explosives	Total
Coal mining:	+ 15.4	+ 21.6	+ 9.5	+ 16.3
Bituminous	+ 16.7	+ 30.3	+ 20.2	+ 20.9
Anthracite	+ 3.9	+ 5.4	+ 3.8	+ 4.4
Metal mining	- 27.3	- 9.5	+ 37.3	+ 36.5
Quarrying and non- metallic mineral mining	+ 21.5	+ 76.3	+ 45.5	+ 44.5
Railway and other con- struction work	+ 74.5	+ 70.2	+ 23.9	+ 26.6
All other purposes	- 43.1	+ 250.4	+ 48.8	+ 45.5
Total	+ 18.6	+ 22.2	+ 30.8	+ 27.0

Table 2. - Explosives (excluding exports) manufactured in the United States and used in the various States in 1936, by use
 GRANULAR BLACK BLASTING POWDER, KEGS OF 25 POUNDS

State	Coal mining	Metal mining	Quarrying and nonmetallic mineral mining	Railway and other construction work	All other purposes	Total
Alabama	20,406	5	1,254	924	1	22,590
Alaska	-	120	-	41	-	161
Arizona	-	855	-	5,329	-	6,184
Arkansas	21,024	-	-	608	5	21,637
California	-	3,736	1,675	208,084	813	214,308
Colorado	9,932	184	3,895	1,356	124	15,491
Connecticut	-	-	130	30	123	283
Delaware	-	-	4	-	-	4
Florida	-	-	8,801	1	-	8,802
Georgia	-	1	2,321	3,710	20	6,052
Idaho	-	40	22	1,986	11	2,059
Illinois	330,039	150	509	1/ 137	64	330,625
Indiana	93,095	-	903	140	19	94,157
Iowa	59,277	-	265	836	826	61,204
Kansas	27,465	3	1,085	2,529	90	31,172
Kentucky	131,520	-	228	2,549	460	134,757
Louisiana	-	-	2	-	-	2
Maine	-	-	294	35	10	339
Maryland & D.C.	6,266	-	566	1,296	1	8,129
Massachusetts	-	-	281	208	1,326	1,815
Michigan	143	-	800	6	1	950
Minnesota	228	9,312	370	968	1,055	11,933
Mississippi	-	-	26	42	-	68
Missouri	32,715	55	4,677	1,462	39	38,948
Montana	26,348	297	1,151	1,361	129	29,286
Nebraska	108	-	973	1,097	759	2,937
Nevada	-	410	-	400	-	810
New Hampshire	-	-	47	11	2	60
New Jersey	-	-	86	5,971	483	6,540
New Mexico	368	-	1,625	738	12	2,743
New York	-	-	882	868	303	2,053
North Carolina	-	2	746	1,010	7	1,765
North Dakota	32,713	-	-	2	48	32,763
Ohio	120,591	-	5,928	3,613	217	130,349
Oklahoma	34,220	9	444	6,510	19	41,202
Oregon	-	2	40	6,095	353	6,490
Pennsylvania:		167	18,663	1,236	332	190,123
Anthracite	76,768	-	-	-	-	-
Bituminous	92,957	-	-	-	-	-
Rhode Island	-	-	60	20	-	80
South Carolina	-	-	294	213	4	511
South Dakota	1,073	-	730	4	162	1,969
Tennessee	25,123	-	804	5,917	4	31,848
Texas	8,218	23	3,505	4,577	50	16,373
Utah	890	1,218	10	4,110	-	6,228
Vermont	-	-	3,476	4	-	3,480
Virginia	15,783	48	2,353	6,396	30	24,610
Washington	2,383	2	2,208	10,365	192	15,150
West Virginia	30,175	-	843	9,998	8	41,024
Wisconsin	-	1/ 22	739	1,663	231	2,611
Wyoming	5,280	2,484	3,765	2,600	6	14,135
Total, 1936	1,205,108	19,101	77,480	305,782	8,339	1,616,810
1935	1,092,897	26,062	64,906	174,934	10,106	1,368,905

1/ Sold by manufacturer but later returned unused to him.

TABLE 3.- Explosives (excluding exports) manufactured in United States and used in the various States in 1936, by use R. I. 3350

PELLET BLACK BLASTING POWDER, CASES OF 25 POUNDS

State	Coal mining	Metal mining	Quarrying and nonmetallic mineral mining	Railway and other construction work	All other purposes	Total
Alabama	36,006	-	292	2	-	36,300
Alaska	-	-	-	-	-	-
Arizona	-	-	-	-	-	-
Arkansas	8,615	-	32	-	-	8,647
California	-	-	-	-	-	-
Colorado	30,160	44	-	-	6	30,210
Connecticut	-	-	-	-	-	-
Delaware	-	-	-	-	-	-
Florida	-	-	-	-	-	-
Georgia	-	-	-	-	-	-
Idaho	-	-	-	-	-	-
Illinois	304,927	108	1/ 1,381	1/ 422	1/ 40	303,192
Indiana	99,369	-	2	152	12	99,535
Iowa	95,657	-	146	-	1/ 78	95,725
Kansas	55,619	128	8	62	-	55,817
Kentucky	160,379	-	5,528	228	-	166,135
Louisiana	-	-	-	-	-	-
Maine	-	-	-	-	-	-
Maryland & D.C.	8,244	-	242	638	-	9,124
Massachusetts	-	-	8	-	-	8
Michigan	5,692	-	20	1	-	5,713
Minnesota	-	-	-	-	-	-
Mississippi	-	-	-	-	-	-
Missouri	21,580	1/18	18	8	81	21,669
Montana	22,506	-	-	-	-	22,506
Nebraska	-	-	-	-	-	-
Nevada	-	-	-	-	-	-
New Hampshire	-	-	-	-	-	-
New Jersey	-	-	3	2	2	7
New Mexico	452	-	-	-	-	452
New York	-	-	2	78	-	80
North Carolina	-	-	32	-	-	32
North Dakota	6,539	-	-	16	-	6,555
Ohio	133,231	-	4,481	126	-	137,838
Oklahoma	11,653	1/32	-	-	-	11,621
Oregon	-	-	-	-	-	-
Pennsylvania:	-	1	2,146	358	23	325,880
Anthracite	194,032	-	-	-	-	-
Bituminous	129,320	-	-	-	-	-
Rhode Island	-	-	-	-	-	-
South Carolina	-	-	-	-	-	-
South Dakota	-	-	-	-	14	14
Tennessee	31,230	-	140	164	-	31,534
Texas	240	-	-	-	-	240
Utah	-	-	-	-	-	-
Vermont	-	-	40	-	-	40
Virginia	39,152	-	-	-	-	39,152
Washington	-	-	-	-	-	-
West Virginia	230,279	-	-	272	-	230,551
Wisconsin	-	-	-	40	-	40
Wyoming	12,460	-	48	-	-	12,508
Total, 1936	1,637,342	231	11,807	1,725	20	1,651,125
1935	1,371,013	536	8,600	1,877	4,576	1,386,602

1/ Sold by manufacturer but later returned unused to him.

TABLE 4. - Explosives (excluding exports) manufactured in the United States and used in the various States in 1936, by use

PERMISSIBLE EXPLOSIVES, POUNDS

State	Coal mining	Metal mining	Quarrying and nonmetallic mineral mining	Railway and other construction work	All other purposes	Total
Alabama	2,242,475	-	2,550	2,500	24,565	2,272,090
Alaska	113,650	-	-	-	-	113,650
Arizona	7,500	-	-	-	-	7,500
Arkansas	32,290	26,600	-	-	-	58,890
California	-	12,600	9,000	6,650	-	28,250
Colorado	434,050	9,350	3,800	17,250	2,900	467,350
Connecticut	-	-	700	-	-	700
Delaware	-	-	-	-	-	-
Florida	-	-	1,900	-	-	1,900
Georgia	2,400	-	-	-	-	2,400
Idaho	200	-	-	-	-	200
Illinois	2,693,752	6,125	42,920	1/ 507	1,840	2,744,130
Indiana	1,243,925	-	1,200	850	1/ 525	1,245,450
Iowa	750	-	-	-	-	750
Kansas	95,700	-	22,200	-	100	118,000
Kentucky	2,566,030	-	6,000	1,350	-	2,573,380
Louisiana	-	-	20,950	-	-	20,950
Maine	-	-	-	-	-	-
Maryland & D.C.	65,400	-	250	750	600	67,000
Massachusetts	-	-	-	-	-	-
Michigan	-	-	6,500	-	-	6,500
Minnesota	-	-	-	-	-	-
Mississippi	-	-	-	-	-	-
Missouri	10,700	-	23,750	50	100	34,600
Montana	13,750	-	-	-	-	13,750
Nebraska	-	-	-	-	-	-
Nevada	-	-	-	-	-	-
New Hampshire	-	-	-	-	-	-
New Jersey	-	-	-	-	-	-
New Mexico	174,400	-	216,050	-	-	390,450
New York	-	-	2,500	-	-	2,500
North Carolina	350	-	350	-	-	700
North Dakota	19,050	-	-	650	-	19,700
Ohio	209,400	-	550	-	3,050	213,000
Oklahoma	10,000	-	-	-	-	10,000
Oregon	-	-	-	-	-	-
Pennsylvania:		1,000	76,500	95,550	11,487	24,783,252
Anthracite	14,271,538	-	-	-	-	-
Bituminous	10,327,177	-	-	-	-	-
Rhode Island	-	-	-	-	-	-
South Carolina	-	-	-	-	-	-
South Dakota	-	10,850	-	-	-	10,850
Tennessee	166,100	-	200	250	-	166,550
Texas	3,150	1,450	114,800	-	-	119,400
Utah	827,500	-	-	7,000	-	834,500
Vermont	-	-	-	-	-	-
Virginia	1,178,600	-	-	1,200	100	1,179,900
Washington	528,600	-	-	1,550	-	530,150
West Virginia	8,790,277	-	27,000	550	29,400	8,847,227
Wisconsin	-	-	-	500	-	500
Wyoming	972,900	-	-	-	-	972,900
Total, 1936	47,001,614	67,975	579,670	136,143	73,617	47,859,019
1935	38,665,149	75,125	328,750	79,980	21,012	39,170,016

1/ Sold by manufacturer but later returned unused to him.

TABLE 5. - Explosives (excluding exports) manufactured in the United States and used in the various States in 1936, by use
HIGH EXPLOSIVES OTHER THAN PERMISSIBLES, POUNDS

R. I. 3350

State	Coal mining	Metal mining	Quarrying and nonmetallic mineral mining	Railway and other construction work	All other purposes	Total
Alabama	199,375	2,924,700	620,782	1,081,800	430,125	5,256,782
Alaska	400	2,227,200	71,400	294,000	16,350	2,609,350
Arizona	-	10,240,889	10,400	1,873,495	61,900	12,186,684
Arkansas	317,400	289,250	252,850	1,371,275	116,399	2,347,174
California	-	8,071,758	2,159,825	14,966,583	253,150	25,451,316
Colorado	60,681	6,114,158	202,135	3,032,910	72,464	9,482,348
Connecticut	-	-	546,080	850,569	54,306	1,342,343
Delaware	-	-	23,025	122,800	6,300	152,125
Florida	-	-	696,240	1,417,440	194,400	2,308,080
Georgia	500	9,300	563,156	676,320	81,425	1,330,701
Idaho	-	2,357,905	51,300	2,719,958	132,012	5,261,175
Illinois	1,746,759	12,600	3,349,371	1,496,952	563,205	7,168,887
Indiana	1,752,569	-	1,128,291	677,467	442,282	4,000,609
Iowa	82,400	-	1,230,063	1,534,673	738,009	3,585,145
Kansas	708,106	1,742,625	807,692	1,033,687	120,124	4,412,234
Kentucky	344,200	-	1,274,075	3,242,960	266,472	5,127,707
Louisiana	-	-	386,500	1,946,776	943,172	3,276,448
Maine	-	-	138,850	626,150	82,200	847,200
Maryland & D.C.	17,455	-	661,220	565,860	24,238	1,268,773
Massachusetts	-	-	505,390	548,469	55,472	1,109,331
Michigan	33,200	6,907,250	2,757,568	1,173,075	296,785	11,167,878
Minnesota	-	7,000,350	61,087	1,300,538	598,200	8,960,175
Mississippi	-	-	8,150	1,000,535	864,175	1,872,860
Missouri	1,006,835	2,269,950	2,030,735	2,403,712	223,082	7,934,314
Montana	260,750	4,825,465	155,050	1,040,510	160,125	6,441,900
Nebraska	-	-	258,359	249,700	108,825	616,884
Nevada	-	3,721,825	74,050	336,525	8,500	4,140,900
New Hampshire	-	-	33,040	289,470	34,090	356,600
New Jersey	-	568,400	890,783	380,907	42,770	1,882,860
New Mexico	27,950	779,175	295,950	1,912,925	89,611	3,105,611
New York	-	2,208,875	4,164,173	4,092,886	183,795	10,649,729
North Carolina	150	59,580	822,732	1,780,948	125,892	2,789,302
North Dakota	141,050	-	-	24,025	27,150	192,225
Ohio	921,233	-	5,292,795	1,682,565	301,604	8,198,197
Oklahoma	71,038	2,741,735	884,000	2,181,972	775,948	6,654,693
Oregon	-	250,815	28,964	6,255,652	512,753	7,048,184
Pennsylvania:		430,425	10,321,830	2,933,548	528,402	29,711,453
Anthracite	14,748,796	-	-	-	-	-
Bituminous	748,452	-	-	-	-	-
Rhode Island	-	-	25,990	108,010	4,000	138,000
South Carolina	-	10,200	249,150	226,700	5,500	491,550
South Dakota	8,000	2,011,141	67,950	124,434	95,800	2,307,325
Tennessee	96,800	931,675	689,507	2,630,575	179,225	4,527,782
Texas	7,300	373,153	1,268,338	4,566,062	585,917	6,800,770
Utah	48,010	5,534,835	75,500	883,733	1,650	6,543,728
Vermont	-	-	202,710	338,260	41,720	499,250
Virginia	171,300	271,350	1,721,111	3,202,867	210,075	5,576,703
Washington	36,752	487,706	417,875	9,800,061	1,219,923	11,962,317
West Virginia	432,250	-	1,571,090	3,733,314	257,958	5,994,612
Wisconsin	-	1,059,305	587,124	2,647,949	785,232	5,079,610
Wyoming	63,600	223,558	144,250	1,391,592	54,282	1,877,282
Total, 1936	24,053,311	76,657,153	49,778,506	98,773,194	12,784,942	262,047,106
1935	21,957,957	55,840,531	34,221,546	79,709,348	8,594,177	200,323,559

1/ Sold by manufacturer but later returned unused to him.

ACKNOWLEDGMENTS

The compilation of this report is made possible by the cooperation of explosives manufacturers and of C. Stewart Comeaux, secretary of the Institute of Makers of Explosives. For this cooperation the Bureau of Mines expresses its deep appreciation.

CLASSIFICATION OF EXPLOSIVES

The explosives discussed in this publication are classified as follows:

1. Black blasting powder. - All black powder with sodium or potassium nitrate as a constituent is classified as black blasting powder.
2. High explosives. - Dynamite and all other high explosives having various trade names and compositions, except permissible explosives, are classified as high explosives.
3. Permissible explosives. - Permissible explosives include ammonium-nitrate explosives, hydrated explosives, organic nitrate explosives, and certain nitroglycerin explosives that contain an excess of free water or carbon. All permissible explosives have passed certain tests of the Bureau of Mines, but they are not to be regarded as permissible unless used in the manner specified by the Bureau.

UNITS OF MEASUREMENT

Throughout this publication black blasting powder, when not expressed in pounds, is in "kegs" of 25 pounds each for granular powder and in "cases" of 25 pounds each for pellet powder. Permissibles and other high explosives invariably are expressed in pounds.

MATERIALS USED IN EXPLOSIVES CONSUMED IN THE UNITED STATES

Manufacturers' reports to the Bureau of Mines covering sales of explosives do not show the quantities of the various constituents used in their manufacture. However, it is interesting to know approximately how much sulphur, sodium nitrate, nitroglycerin, and other materials are consumed in the manufacture of the explosives used for industrial purposes in the United States. The quantities of such materials can be estimated by analyzing the figures showing the quantities of explosives sold.

A summary of the sales statistics for 1936 that form the basis of the present publication was furnished to J. E. Crawshaw, explosives engineer of the Pittsburgh Experiment Station of the Bureau. Crawshaw estimated therefrom the quantities of the various materials used in the manufacture of explosives. The results of this estimate are shown in table 6.

TABLE 6. - Estimated quantities of certain materials used in explosives consumed in the United States during 1936, pounds

Materials	Black powder		Permissi- bles	Other high explosives	Total
	Granular	Pellet			
Sulphur	5,255,000	5,232,000	-	5,440,000	15,927,000
Charcoal	6,063,000	6,037,000	-	-	12,100,000
Sodium nitrate	29,103,000	28,977,000	2,846,000	105,481,000	166,407,000
Nitroglycerin	-	-	4,598,000	77,275,000	1/ 81,873,000
Nitrotoluenes	-	-	-	1,686,000	1,686,000
Nitrocellulose	-	-	-	871,000	871,000
Ammonium nitrate	-	-	31,311,000	27,932,000	59,243,000
Antacid	-	-	219,000	1,852,000	2,071,000
Wood pulp	-	-	3,120,000	16,073,000	19,193,000
Paper	-	619,000	2,034,000	10,731,000	13,384,000
Paraffin	-	413,000	2,034,000	7,796,000	10,243,000
Total	40,421,000	41,278,000	46,162,000	255,137,000	382,998,000
Absorbents other than wood pulp, moisture, and other compounds	-	-	1,697,000	6,910,000	8,607,000
Total sales	40,421,000	41,278,000	47,859,000	262,047,000	391,605,000

1/ Nitroglycerin contains nitroglycerin, nitropolyglycerin, nitrosugar, and ethylene glycol dinitrate (approximately 21,000,000 pounds in 1936).

Table 7, also prepared by Mr. Crawshaw, shows the quantity of nitrogen materials contained in the explosives consumed in the United States during 1936.

TABLE 7. - Nitrogen materials used in explosives consumed in the United States during 1936

	Pounds	Percent nitrogen	Pounds, nitrogen	Pounds, nitric acid 100 percent	Pounds, sodium nitrate, 100 percent	Pounds, sodium nitrate, commercial
Nitroglycerin	81,872,900	18.5	15,146,490	68,159,190	91,960,380	99,225,250
Nitrotoluenes	1,685,410	15.4	259,550	1,167,990	1,575,850	1,700,340
Nitrocellulose	871,330	12.0	104,560	470,520	634,830	684,980
Sodium nitrate	166,407,400	16.5	27,407,300	-	166,407,400	169,735,390
Ammonium nitrate 1/2/	59,243,030	35.0	20,735,060	46,653,890	62,945,430	67,918,120
Total	-	-	63,652,960	116,451,590	323,523,890	339,264,080

1/ Contains approximately 21,000,000 pounds ethylene glycol dinitrate.

2/ Pounds ammonia, 12,589,140.

CAPACITY OF EXPLOSIVES MILLS

Table 8 shows the percentage of manufacturing capacity that was utilized at explosives mills in 1936 and previous years.

Mills at which black blasting powder was manufactured were operated to 42.3 percent of their rated capacity during 1936. This record represented a larger utilization of full mill capacity than in any year during the period 1932-36 for which reports covering all mills are available, and it also represents a larger utilization of mill capacity than in any year since 1927 for those mills furnishing reports of capacity. Reports relating to manufacturing capacity were not furnished by certain mills previous to 1932. In comparing the yearly percentage of utilization of manufacturing capacity of the mills, it should be borne in mind that the total capacity for the mills may change from year to year and that the percentage of utilization in any given year is relative to the rated capacity of the mills for that year. In 1936 the rated capacity of black-powder mills was about 30 percent less than it was in 1932.

Mills engaged in the manufacture of permissibles and other high explosives were operated to 55.7 percent of their rated capacity in 1936. This figure does not represent all mills that were active in 1936 but only those that furnished reports of their capacity to the Bureau of Mines; these represented about 90 percent of the entire industry. The percentage of utilization of manufacturing capacity in 1936 was the highest since 1930. As far as figures are available for the years since 1922, and which represent varying degrees of coverage of the industry from year to year, the percentage of utilization of capacity has ranged from 28.6 in 1932 to 68.0 in 1926. In 1926 the total quantity of explosives that could have been produced by the mills that reported their capacity for that year was 471 million pounds; compared with this, the mills that reported their capacity for 1932 could have produced 536 million pounds. A maximum production of nearly 527 million pounds could have been manufactured by mills that reported their capacity for 1936.

As previously stated, the percentage of utilization of mill capacity shown in table 8 should be interpreted in relation to the manufacturing capacity reported for the respective years. The proportion of the capacity utilized during 1936 is shown graphically for certain individual mills in figures 1 and 2.

THE HISTORY OF THE UNITED STATES

The first part of the history of the United States is the period from the discovery of the continent by Christopher Columbus in 1492 to the establishment of the first permanent settlements. This period is characterized by the exploration of the continent by Spanish, French, and English explorers, and the establishment of the first permanent settlements by the English in 1607. The second part of the history is the period from the establishment of the first permanent settlements to the American Revolution in 1776. This period is characterized by the growth of the colonies, the struggle for independence, and the establishment of the United States as a new nation.

The third part of the history is the period from the American Revolution to the Civil War in 1861. This period is characterized by the growth of the United States, the struggle for slavery, and the establishment of the United States as a new nation. The fourth part of the history is the period from the Civil War to the present. This period is characterized by the growth of the United States, the struggle for civil rights, and the establishment of the United States as a new nation.

The fifth part of the history is the period from the present to the future. This period is characterized by the growth of the United States, the struggle for civil rights, and the establishment of the United States as a new nation.

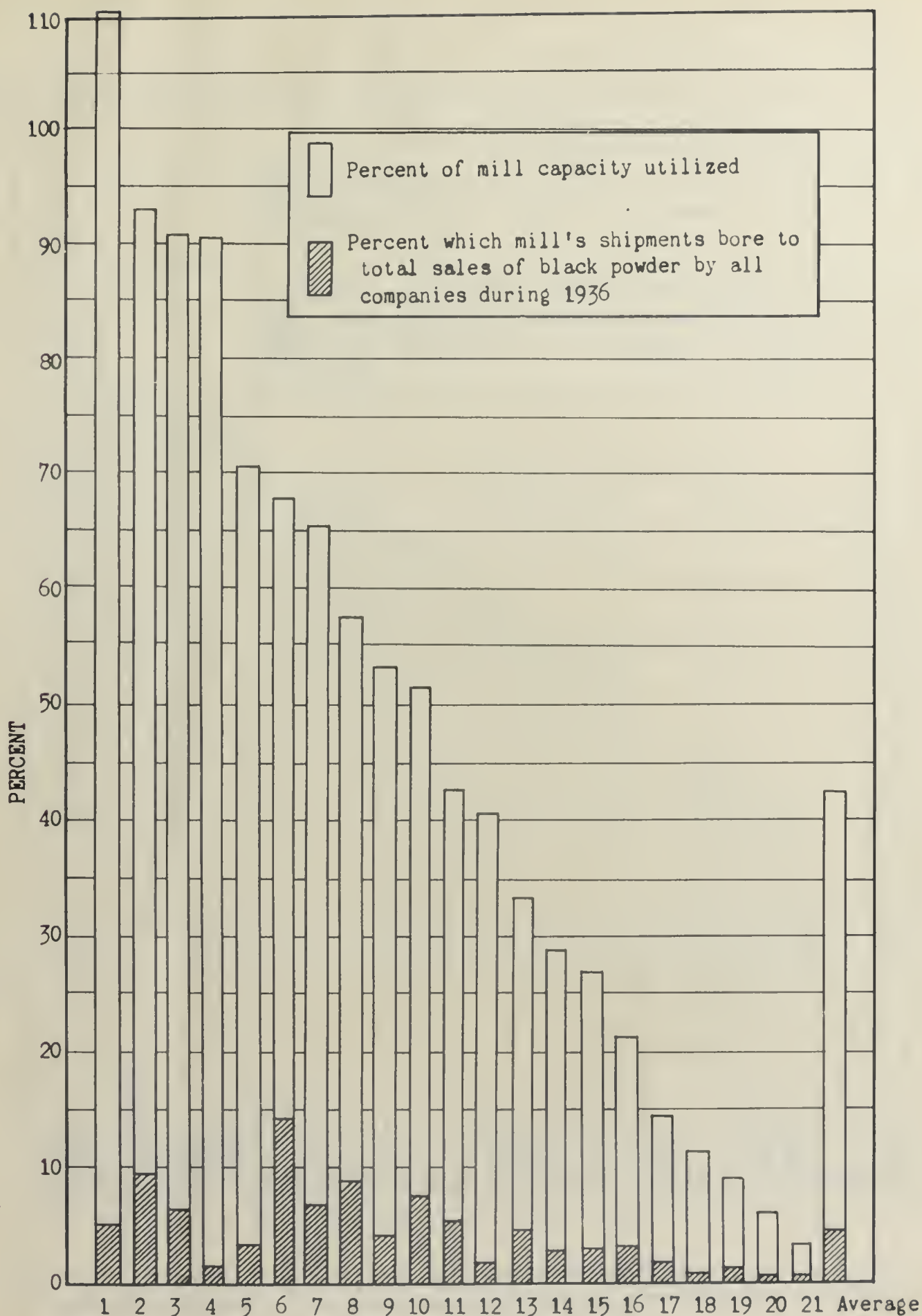


Figure 1.— Percent of utilization of manufacturing capacity of mills making black blasting powder during the calendar year 1936. (Each upright bar represents an individual mill.)

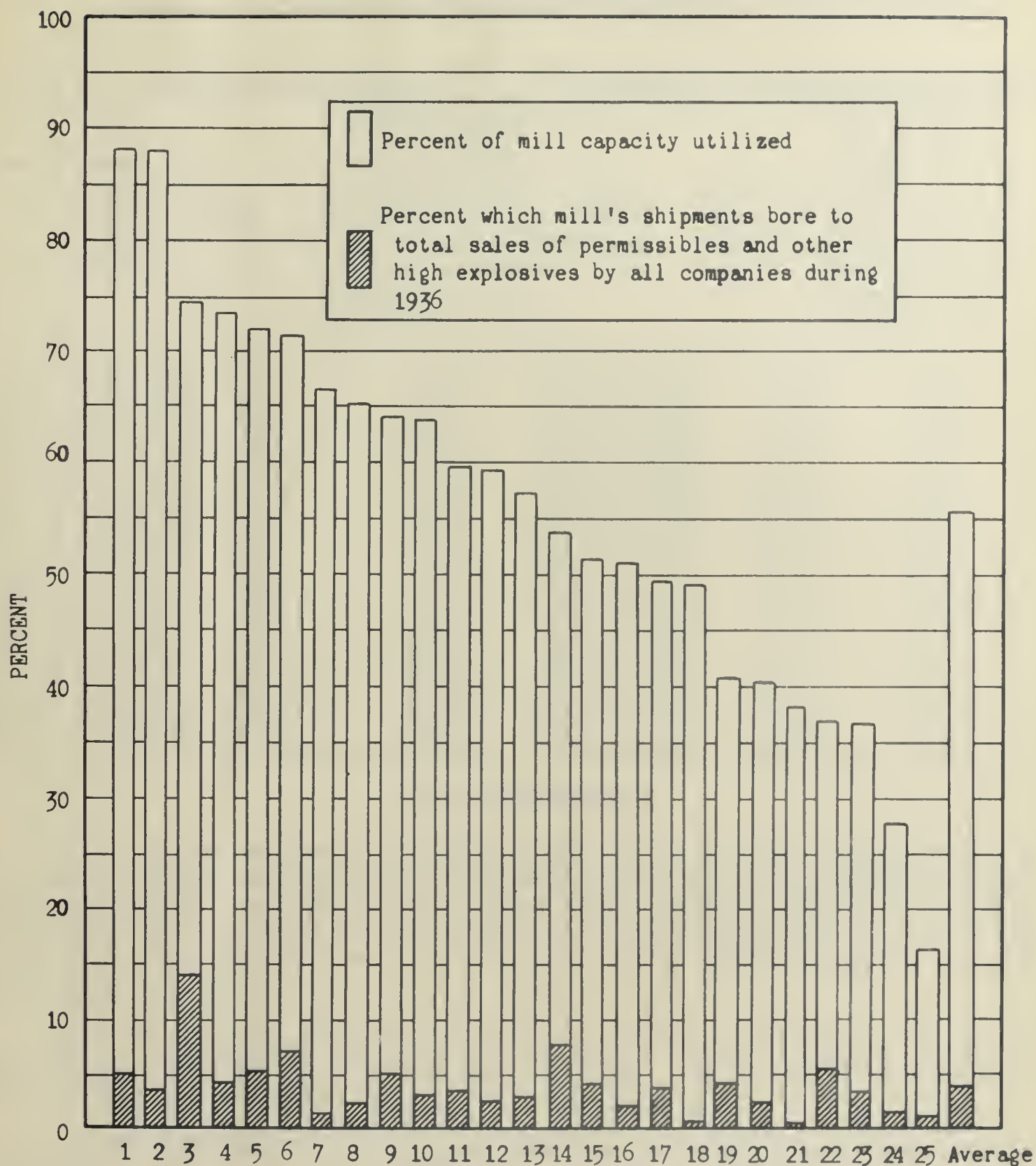


Figure 2.— Percent of utilization of manufacturing capacity of mills making permissibles and other high explosives during the calendar year 1936.
(Each upright bar represents an individual mill.)

TABLE 8. - Percentage utilization of manufacturing capacity of mills making black blasting powder or high explosives, 1922-36, by years

Year	Black blasting powder			Permissibles and other high explosives		
	Year's capacity (kegs or cases of 25 pounds)	Percentage utilized	Percentage of total sales of entire industry represented by sales of mills reporting capacity as indicated in this table	Year's capacity (pounds)	Percentage utilized	Percentage of total sales of entire industry represented by sales of mills reporting capacity as indicated in this table
1922	12,465,000	42.4	73.6	460,116,000	51.5	85.2
1923	12,635,000	50.6	76.9	446,490,000	63.0	81.4
1924	12,364,000	40.9	75.1	456,512,000	62.2	82.3
1925	12,180,000	38.5	73.7	468,590,000	62.3	80.4
1926	11,443,000	40.5	71.6	471,340,000	68.0	80.3
1927	9,553,000	43.3	75.6	480,590,000	63.5	79.2
1928	9,420,000	42.3	78.8	498,340,000	59.8	78.9
1929	9,820,000	39.2	82.7	525,840,000	65.1	84.2
1930	9,315,000	33.7	82.5	524,840,000	57.9	84.8
1931	11,020,000	28.5	99.9	535,350,000	42.9	88.4
1932	10,190,000	24.3	100.0	536,300,000	28.6	90.7
1933	9,645,000	27.0	100.0	499,300,000	34.8	88.1
1934	7,235,000	38.9	100.0	520,010,000	44.6	91.3
1935	7,295,000	37.6	100.0	521,350,000	43.9	90.4
1936	7,064,000	42.3	100.0	526,750,000	55.7	89.9

USES OF EXPLOSIVES

Table 9 shows the percentage distribution of the 1935 and 1936 explosives production according to various uses of the explosives. While the coal industry consumed the bulk of the black blasting powder and permissible explosives produced during 1936, railway and other construction work used the largest portion of high explosives other than permissibles, with metal mining ranking second in quantity of high explosives used.

Table 10 gives the quantities of the three classes of explosives consumed by the various industries in which explosives are used. Table 10 covers the years 1930, 1934, 1935, and 1936.

The total production of explosives manufactured in the United States during each year of the 25-year period 1912-1936, expressed in pounds for each class of explosives, is shown in table 11.



TABLE 9. - Percentage distribution of total sales of each class of explosives, by uses, 1935 and 1936

Kind of explosives	Coal mining			Metal mining	Quarrying and non-metallic mineral mining	Railway and other construction work	All other purposes	Total
	Anthracite	Bituminous	Total					
1935								
Black blasting powder	9.5	79.9	89.4	1.0	2.7	6.4	0.5	100.0
High explosives	7.1	3.8	10.9	27.9	17.1	39.8	4.3	100.0
Permissible explosives	34.6	64.1	98.7	.2	.8	.2	.1	100.0
Total	11.1	28.5	39.6	18.4	11.8	27.3	2.9	100.0
1936								
Black blasting powder	8.3	78.7	87.0	.6	2.7	9.4	.3	100.0
High explosives	5.6	3.6	9.2	29.2	19.0	37.7	4.9	100.0
Permissible explosives	29.8	68.4	98.2	.1	1.2	.3	.2	100.0
Total	9.1	27.2	36.3	19.7	13.5	27.2	3.3	100.0

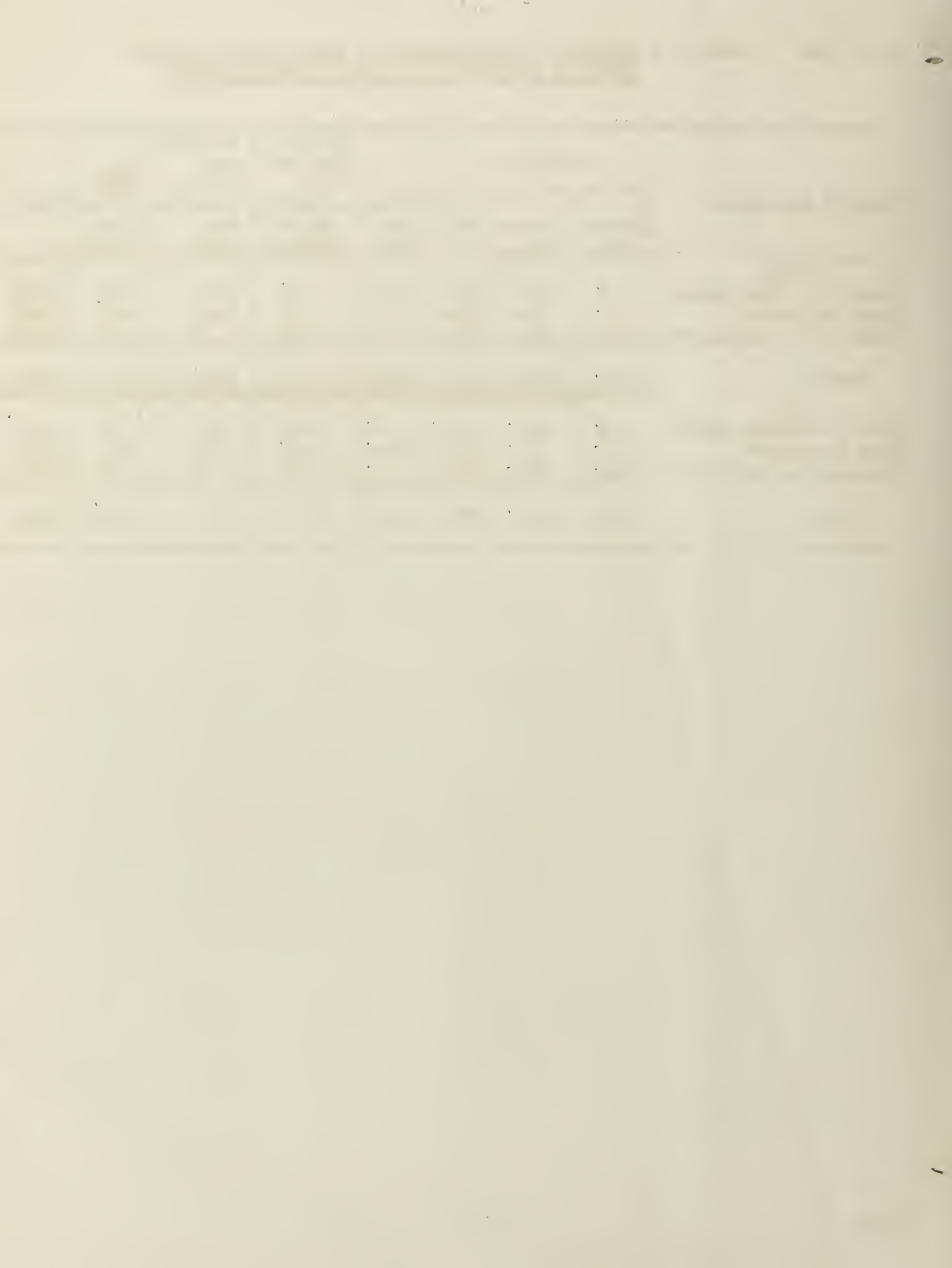


TABLE 10. - Consumption of explosives in the United States
1930 and 1934 to 1936, by classes of consuming industries

(Excluding exports)

Use	1930	1934	1935	1936
Black blasting powder (kegs and cases of 25 pounds):				
Coal mining	3,427,397	2,416,647	2,463,910	2,842,450
Metal mining	55,539	23,977	26,598	19,332
Quarrying and nonmetallic mineral mining	153,022	73,970	73,506	89,287
Railway and other construction work	288,816	232,055	176,811	308,507
All other purposes	70,163	10,753	14,682	8,359
Total	1/ 3,994,937	1/ 2,757,402	1/ 2,755,507	1/ 3,267,935
Black blasting powder(pounds):				
Coal mining	85,684,925	60,416,175	61,597,750	71,061,250
Metal mining	1,388,475	599,425	664,950	483,300
Quarrying and nonmetallic mineral mining	3,825,550	1,849,250	1,837,650	2,232,175
Railway and other construction work	7,220,400	5,801,375	4,420,275	7,712,675
All other purposes	1,754,075	268,825	367,050	208,975
Total	1/99,873,425	1/68,935,050	1/68,887,675	1/81,698,375
High explosives other than permissibles (pounds):				
Coal mining	28,148,326	22,212,202	21,957,957	24,053,311
Metal mining	91,161,051	45,689,578	55,840,531	76,657,153
Quarrying and nonmetallic mineral mining	68,333,964	34,491,023	34,221,546	49,778,506
Railway and other construction work	78,361,721	95,037,665	73,709,348	98,773,194
All other purposes	25,385,865	9,194,559	8,594,177	12,784,942
Total	291,390,927	206,625,027	200,323,559	262,047,106
Permissible explosives(pounds):				
Coal mining	52,378,341	38,769,659	38,665,149	47,001,614
Metal mining	183,294	27,844	75,125	67,975
Quarrying and nonmetallic mineral mining	501,502	286,775	328,750	579,670
Railway and other construction work	213,706	92,756	79,980	136,143
All other purposes	549,303	30,811	21,012	73,617
Total	53,826,146	39,207,845	39,170,016	47,859,019
Grand total (pounds):				
Coal mining	166,211,592	121,398,036	122,220,856	142,116,175
Metal mining	92,732,820	46,316,847	56,580,606	77,208,428
Quarrying and nonmetallic mineral mining	72,661,016	36,627,048	36,387,946	52,590,351
Railway and other construction work	85,795,827	100,931,796	84,209,603	106,622,012
All other purposes	27,689,243	9,494,195	8,982,239	13,067,534
Total	445,090,498	314,767,922	308,381,250	391,604,500

1/ Includes granular powder and pellet powder.

TABLE 11. - Production of explosives in the
United States, 1912-36, pounds

Year	Permissible explosives	High explosives other than permissibles	Black blasting powder		Total
			Granular ^{1/}	Pellet ^{1/}	
1912	24,630,270	234,469,492	230,293,375	-	489,393,137
1913	27,685,770	242,386,544	229,939,525	-	500,011,839
1914	25,697,818	218,453,971	206,099,700	-	450,251,489
1915	27,349,909	235,828,587	197,722,300	-	460,900,796
1916	34,685,240	255,154,787	215,575,025	-	505,415,052
1917	43,040,722	262,316,080	277,118,525	-	582,475,327
1918	46,045,233	206,416,077	246,663,350	-	499,124,660
1919	38,854,832	198,268,588	180,511,050	-	417,634,470
1920	53,962,841	229,112,084	254,879,825	-	537,954,750
1921	41,133,851	170,952,402	160,021,250	-	372,107,503
1922	43,429,768	209,476,084	178,866,225	-	431,772,077
1923	60,371,314	267,405,220	201,951,325	-	529,727,859
1924	55,134,151	273,323,178	167,076,125	-	495,533,454
1925	58,353,217	286,434,529	156,964,100	-	501,751,846
1926	67,685,416	310,517,937	157,686,825	-	535,890,178
1927	63,847,161	303,467,767	131,696,200	-	499,011,128
1928	60,707,757	292,785,010	98,004,375	23,754,000	475,251,142
1929	62,669,434	326,993,276	86,818,225	33,227,425	509,708,360
1930	53,826,146	291,390,927	63,138,550	36,734,875	445,090,498
1931	41,577,607	216,157,246	46,300,450	33,529,600	337,564,903
1932	32,225,193	137,907,607	35,793,200	27,961,300	233,887,300
1933	33,927,443	157,849,273	33,887,225	30,323,450	255,987,391
1934	39,207,845	206,625,027	37,192,700	31,742,350	314,767,922
1935	39,170,016	200,323,559	34,222,625	34,665,050	308,381,250
1936	47,859,019	262,047,106	40,420,250	41,278,125	391,604,500

^{1/} Pellet powder was first manufactured for general sale in the United States in 1925; production prior to 1928 is included with granular powder.

MANUFACTURERS OF EXPLOSIVES

The companies that manufactured explosives in the United States during 1936 are listed below:

Companies manufacturing explosives in 1936

(B, black powder; H, high explosives other than permissibles; P, permissibles)

H, P	American Cyanamid & Chemical Corporation, New York, N. Y.
H	Apache Powder Co., Benson, Ariz.
B, H, P ...	Atlas Powder Co., Wilmington, Del.
B, H, P ...	Austin Powder Co., Cleveland, Ohio.
H, P	Columbia Powder Co., East Alton, Ill.
H	Dixie Torpedo Co., Owensboro, Ky.
B, H, P ...	E. I. du Pont de Nemours & Co., Inc., Wilmington, Del.
B	Egyptian Powder Co., East Alton, Ill.
B	Equitable Powder Manufacturing Co., East Alton, Ill.
H	Grange Powder Co., Seattle, Wash.
H	Halifax Explosives Co., Los Angeles, Calif.
B, H, P ...	Hercules Powder Co., Wilmington, Del.
H, P	Illinois Powder Manufacturing Co., St. Louis, Mo.
H	Independent Explosives Co., Cleveland, Ohio.
B	King Powder Co., Cincinnati, Ohio.
H, P	Liberty Powder Co., Pittsburgh, Pa.
H	Southern Dinite Co., Embreeville, Tenn.
H	Titan Explosives Co., Portland, Oreg.
H	Trojan Powder Co., Allentown, Pa.
B	United States Powder Co., Terre Haute, Ind.
H	West Coast Powder Co., Everett, Wash.
B	Western Powder Manufacturing Co., Peoria, Ill.

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SEPTEMBER 1937

DEPARTMENT OF THE INTERIOR

UNITED STATES BUREAU OF MINES

JOHN W. FINCH, DIRECTOR

REPORT OF INVESTIGATIONS

A METHOD OF DETERMINING POROSITY: A LIST OF POROSITIES
OF OIL SANDS



BY

D. B. TALIAFERRO, JR., T. W. JOHNSON, AND E. J. DEWEES

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A METHOD OF DETERMINING POROSITY: A LIST OF POROSITIES
OF OIL SANDS 1/

By D. B. Taliaferro, Jr., 2/ T. W. Johnson, 3/ and E. J. Dewees 4/

INTRODUCTION

Information regarding the pores or voids in reservoir rocks is vitally important in the scientific extraction of petroleum from natural underground reservoirs, and petroleum technologists concerned with subsurface conditions have devised various methods that give reasonably accurate information as to the porosities of oil and gas-bearing sands.^{5/} In this work, several investigators have made use of the experience of the ceramic and other industries.

The method of determining the effective porosity of oil sands described in this report falls within the general classification of gas-expansion methods. It was devised and developed at the United States Bureau of Mines Petroleum Experiment Station, Bartlesville, Okla., as a part of the research dealing with fluid and energy relations in reservoirs and improved oil-recovery methods. The apparatus and procedure are well adapted to the requirements of the Bureau, and the following detailed description has been prepared so that others may make use of the information. A tabulation at the end of the report gives the results of several hundred porosity determinations for oil sands in fields of 11 States.

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- 1/ The Bureau of Mines will welcome reprinting of this paper, provided the following footnote acknowledgment is used: "Reprinted from U. S. Bureau of Mines Report of Investigations 3352."
 - 2/ Associate petroleum engineer, U. S. Bureau of Mines, Petroleum Experiment Station, Bartlesville, Okla.
 - 3/ Former natural-gas engineer, U. S. Bureau of Mines, Petroleum Experiment Station, Bartlesville, Okla.
 - 4/ Associate petroleum engineer, U. S. Bureau of Mines, Petroleum Experiment Station, Bartlesville, Okla.
 - 5/ Throughout this paper, the term "sand" is used with the meaning customarily implied in discussions of subjects relating to petroleum engineering and geology, namely, an oil-bearing formation, usually a sandstone; but frequently other kinds of porous rocks or unconsolidated sands may be intended. The term "sand grains" is used to refer to the solid material of which the porous rock is composed, regardless of its chemical composition.

In 1922, Washburn and Bunting^{6/} described a gas-expansion method for determining the porosity of ceramic bodies, in which the test specimen is placed in a closed glass vessel that is connected to a second vessel by a glass tube containing a stopcock. With the stopcock closed, the first vessel is evacuated while the second is open to the atmosphere. After the atmospheric pressure and the reduced pressure in the first vessel are measured, the second vessel is closed from the atmosphere. The stopcock between the containers then is opened and the resulting pressure in the two vessels measured. Having previously determined the volume of the two vessels, it is possible to calculate the volume of the sand grains or the material in the test specimen by the application of Boyle's law.

In 1924, B. E. Lindsly, formerly with the Bureau of Mines, made some preliminary experiments with a gas-expansion method to determine not only the volume of sand grains in the specimen but also the bulk or total volume of the specimen. The determination of the bulk volume by this method required sealing the pores in the specimen with paraffin. As a part of the study being made by the Bureau of Mines of oil-sand characteristics, it was desired to determine the porosity of oil-sand specimens used in permeability determinations without contaminating the specimens. The method and apparatus suggested by Lindsly was developed in detail to determine the volume of sand grains in a given specimen. However, the bulk volume of the specimen is determined, as in a number of other methods, by displacement of mercury from a pycnometer.

The method described in this report differs from the Washburn-Bunting method in two respects; first, pressures from 40 to more than 200 pounds per square inch are used rather than atmospheric pressure, and, second, the gas is expanded to a definite pressure (atmospheric pressure) and the volume measured rather than expanded to a definite volume and the pressure measured. Coberly and Stevens^{7/} describe in detail the apparatus and calculations for a gas-expansion method using pressures above atmospheric pressure and expanding to a definite volume.

ACKNOWLEDGMENTS

This report was prepared under the general supervision of R. A. Cattell, chief engineer, Petroleum and Natural-Gas Division, United States Bureau of Mines, and N. A. C. Smith, supervising engineer, Petroleum Experiment Station, United States Bureau of Mines, Bartlesville, Okla., and under the direct supervision of W. B. Berwald, senior petroleum engineer, United States Bureau of Mines, Bartlesville, Okla.

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- ^{6/} Washburn, E. W., and Bunting, E. N., Determination of Porosity by the Method of Gas Expansion: Jour. Am. Ceram. Soc., vol. 5, Feb. 1922, pp. 112-129.
- ^{7/} Coberly, C. J., and Stevens, A. B., Development of Hydrogen Porosimeter: Trans., Am. Inst. Min. and Met. Eng., vol. 103, Petrol. Devel. and Technol., 1933, pp. 261-269; see also Oil Weekly, vol. 68, no. 6, January 23, 1933, pp. 17-22.

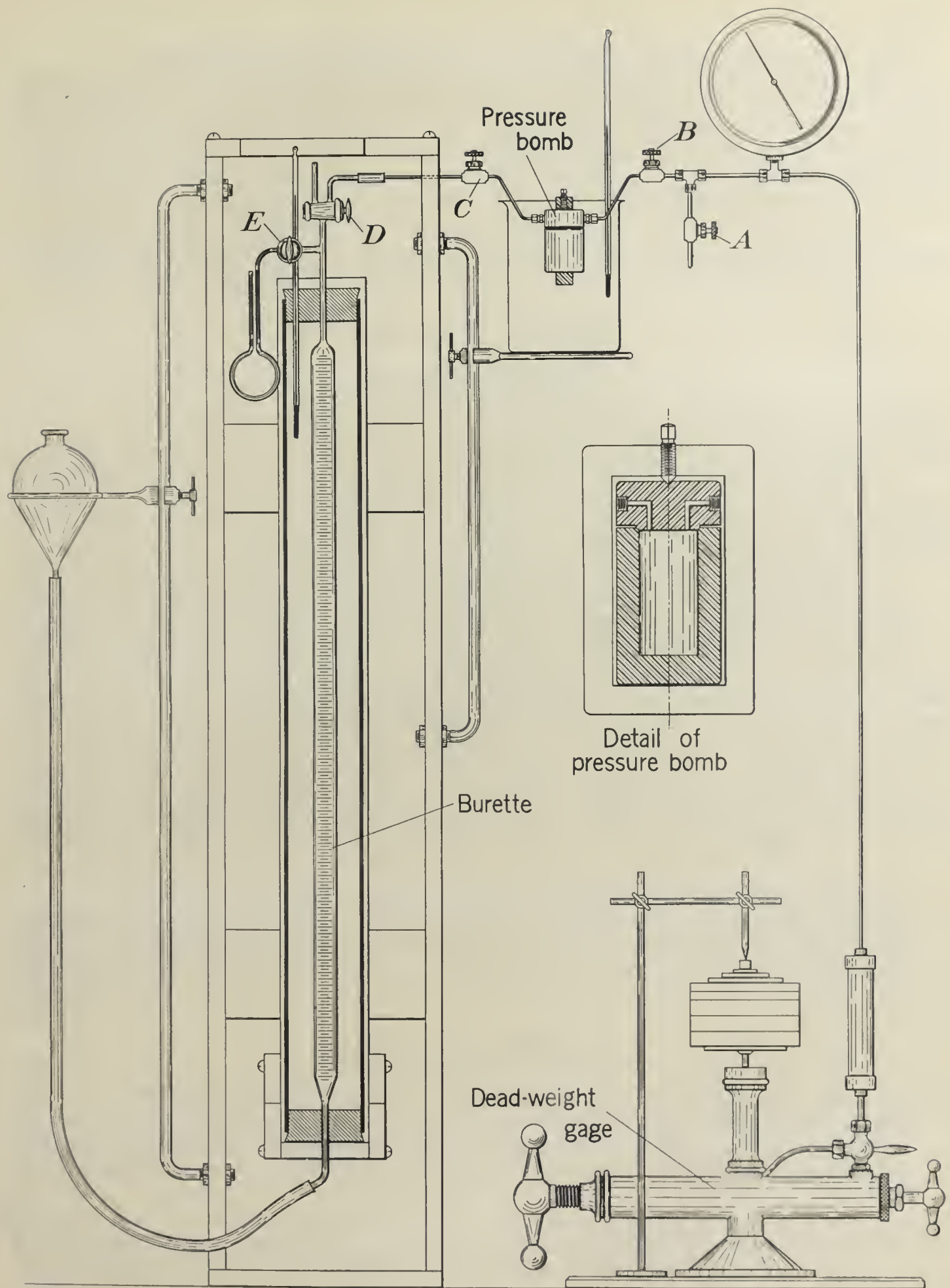


Figure 1.—Apparatus for porosity determinations.

Acknowledgment is made to E. S. Burnett, senior mechanical engineer, Helium Plant, United States Bureau of Mines, Amarillo, Texas, for his assistance in calculating the factors for the deviation of air from Boyle's law and constructive criticism of the manuscript. The authors express their thanks, also, to W. S. Morris, field chairman, East Texas Engineering Association, and to the following members of the Bureau of Mines for reviewing and criticizing the manuscript: H. C. Fowler, W. B. Berwald, Gustav Wade, Ralph H. Espach, C. C. Anderson, Charles B. Carpenter, and Dale Nichols. The drawings were made by J. M. Seward and Ralph Wright, and many of the porosity determinations included in the report were made by G. B. Spencer.

DESCRIPTION OF METHOD

General Description

The volume of the pores in the specimen is determined by subtracting the volume of the sand grains from the bulk volume of the specimen. The volume of the sand grains is determined by noting the reduction of void space in a vessel or bomb of known volume when the specimen is placed in it. The volume of space in the bomb not occupied by the sand grains of a specimen is determined by introducing compressed gas in the bomb, observing the pressure in the bomb, and later releasing the compressed gas and measuring accurately its volume at atmospheric pressure. With the above data, the volume of the void space in the bomb is calculated by use of the pressure-volume relationship expressed by Boyle's law, with a correction factor for the deviation from Boyle's law at the pressure and temperature of the test. When there is a difference in temperature between the compressed gas in the bomb and its expanded volume as measured in the burette, the expanded volume is converted to a volume at the temperature of the gas in the bomb by application of the temperature-volume relationship expressed by Charles' law. The volume of the empty bomb is determined in the same manner. The difference between the volume of the empty bomb and the volume of void space in the bomb with the test specimen in it gives the volume of the sand grains.

The total or bulk volume of the specimen is determined with a pycnometer using mercury. The difference between the bulk volume and the volume of the grains gives the volume of voids in the specimen, and this volume divided by the bulk volume multiplied by 100 gives the porosity in percent.

Description of Apparatus

The apparatus consists of a pressure bomb in which the test specimen is placed, equipment for measuring pressure, and a burette into which the gas is expanded and measured at atmospheric pressure. Figure 1 shows the assembly that includes both a spring gage and a deadweight gage^{8/} for

^{8/} Rawlins, E. L., and Wosk, L. D., Leakage from High-Pressure Natural-Gas Transmission Lines: Bull. 265, Bureau of Mines, 1928, pp. 7-9

measuring pressure. The use of the deadweight gage is desirable, but a spring gage may be used if it is calibrated carefully. The pressure bomb consists of a steel cylinder, 1 3/4 inches outside diameter by 3 1/4 inches long; the corresponding inside dimensions are 1 inch by 1 7/8 inches. The cap or cover of the bomb fits with a ground joint and is held tightly in place with a clamp. As shown under the caption "Accuracy of the Method," it is desirable to have the volume of free space in the bomb as nearly equal as possible to the bulk volume of the test specimen. This is accomplished by inserting in the bomb a number of steel disks having approximately the same diameter as the inside of the bomb. If the specimen is less than 1 inch in diameter, steel collars, slugs, or any other solid material may be used to reduce the void space in addition to the disks. If irregular-shaped specimens are used, the void space may be reduced by filling in around the specimen with glass beads, steel balls, or other similar material. The collars, disks, and slugs should be numbered and a record made of those used in each determination, so that whenever the same combinations are used on subsequent tests the previous determination of the volume of the bomb may be used. By following this procedure, it becomes possible to fit any specimen in the bomb with a combination that has been used previously. The burette is of a good grade, graduated in tenths of milliliters, and has a capacity of 100 milliliters. Both the bomb and burette are operated in water baths in order to prevent any rapid change of their temperature and to facilitate measurement of the temperature. Valves A, B, and C are small brass needle valves, and D and E are glass stopcocks.

Method of Operation

Having reduced the volume of the bomb with steel collars or other solid material, making it as nearly equal as possible to the bulk volume of the specimen, the next step is to determine this reduced volume of the bomb. It is determined most accurately by using the highest pressure possible without exceeding the capacity of the burette when the air is expanded to atmospheric pressure. After an estimate is made of the highest pressure that may be used, the proper weights are placed on the deadweight gage, valve C is closed, valve B is opened, and dry air is admitted to the system through valve A until the weights on the gage are in a floating position. Valve A then is closed and the pressure in the system allowed to come to equilibrium. After the pressure, the bomb temperature, and the position of the mercury level in the burette are recorded, valve B is closed. By adjusting stopcock D to the proper position and opening valve C, the air then is allowed to expand into the burette with stopcock E closed. The pressure in both the bomb and burette are brought to atmospheric pressure by adjusting the level of the mercury in the leveling bulb to approximately that in the burette, turning stopcock E, which connects the burette to the small water-filled manometer, and then making any necessary adjustment to bring the manometer into balance against atmospheric pressure. The mercury level in the burette and the burette temperature then are recorded. With these data, the capacity of the pressure bomb is calculated. (See "Method of Calculation").

The next step is to place the test specimen in the bomb and determine the volume of void space remaining in the bomb by following the same

procedure. This volume will include the volume of the pores in the specimen as well as the space in the bomb not occupied by the specimen.

The third step in the determination is to find the bulk volume, or the volume of the sand grains plus the volume of the pore space in the specimen. This is done by displacing mercury from a pycnometer and weighing the displaced mercury.

A wide-mouth pycnometer is used, $1\frac{1}{4}$ inches in diameter and 2 inches deep. The stopper fits with a ground joint and has a hole through it about 2 mm in diameter. The pycnometer is filled with clean mercury and the stopper is pressed firmly into place, causing the excess mercury to overflow. The mercury in the hole in the stopper should be wiped off flush with the surface of the stopper, and any particles of mercury clinging to the pycnometer should be brushed off with a camel's-hair brush. The pycnometer then is placed in a clean container, the stopper removed, and the specimen placed on the surface of the mercury. If mercury is displaced by pushing the specimen down with the stopper, an air-bubble clings to the specimen at the point where it touches the stopper, causing an appreciable error in the volume determination. This condition is avoided by running a piece of piano wire through the hole in the stopper and holding the specimen below the surface of the mercury until all the air is expelled from the pycnometer. The wire is removed before the stopper is pressed down firmly, thereby assuring sufficient mercury in the pycnometer to fill the hole in the stopper. The pycnometer is brushed clean, as before, and the displaced mercury carefully transferred to a weighing bottle or other suitable container and weighed. The bulk volume of the specimen is obtained by multiplying the weight of the displaced mercury, measured in grams, by its specific volume at the temperature of the test. Table 1 gives the volume, in milliliters, of 1 gram of mercury at several temperatures.

TABLE 1.- Specific volume of mercury (metric units)

65°F. =	0.073799	85°F. =	0.073948
70°F. =	.073836	90°F. =	.073985
75°F. =	.073874	95°F. =	.074022
80°F. =	.073911	100°F. =	.074059

If the pores in the test specimen are large enough to allow mercury to enter them, the core should be saturated with some liquid before it is placed in the pycnometer. Tetrachloroethane has been found satisfactory for this purpose.

Method of Calculation

The calculation of the volume of the empty bomb is based on the pressure-volume-temperature relationship for gases expressed by the ideal gas laws or,

$$\frac{P_B V_B}{T_B} = \frac{P_b V_1}{T_1} \text{-----} \quad (1)$$

where,

P_B = pressure of the gas in the bomb, lb. per sq. in. abs.,

T_B = temperature of the gas in the bomb, °F. abs.,

V_B = volume of the bomb, milliliters,

P_b = pressure of the gas in the burette, lb. per sq. in. abs.
(also pressure of the gas in the bomb after expansion of the gas).

T_1 = temperature of the gas after expansion, °F. abs.,

V_1 = total volume of the gas after expansion, milliliters.

The total volume of gas after expansion is equal to the volume of gas measured in the burette plus the volume of gas remaining in the bomb, or,

$$\frac{P_b V_1}{T_1} = \frac{P_b V_b}{T_b} + \frac{P_b V_B}{T_B} \text{-----} \quad (2)$$

where,

V_b = volume of gas measured in the burette, millimeters,

T_b = Temperature of gas measured in the burette, °F. abs., and other symbols as previously defined.

Substituting this value of $\frac{P_b V_1}{T_1}$ in equation 1,

$$\frac{P_B V_B}{T_B} = \frac{P_b V_b}{T_b} + \frac{P_b V_B}{T_B} \text{-----} \quad (3)$$

Many common gases conform approximately to Boyle's law at ordinary ranges of pressure and temperature, but no real gas follows it exactly. The porosity determinations given in this report were made with dry compressed

air. Within the ranges of pressure and temperature used in these experiments, air with an increase in pressure is more compressible; and conversely, with a decrease in pressure is more expansible than the ideal gas that follows Boyle's law. To obtain the true value of pressure times volume ($P_s V_s$) at standard pressure (P_s), the product of the pressure times volume at a given or initial condition $P_x V_x$ is multiplied by a factor (y), which, for air under the conditions of pressure and temperature of these experiments, is greater than one. When a correction factor for the deviation from Boyle's law is applied, equation 3 becomes,

$$y \frac{P_B V_B}{T_B} = \frac{P_b V_b}{T_b} + \frac{P_b V_B}{T_B} \text{ --- (4)}$$

where y is the deviation factor at pressure P_B , letting the value of y at P_b equal 1 from which,

$$V_B \frac{yP_B - P_b}{T_B} = \frac{P_b V_b}{T_b} \text{ --- (5)}$$

and

$$V_B = \frac{T_B}{T_b} = \frac{V_b P_b}{yP_B - P_b} \text{ --- (6)}$$

The above formulas are applicable in determining the void space in the bomb with the specimen in it. The term V_B then is defined as the volume of void space in the bomb with the specimen in it instead of the volume of the empty bomb.

The following example illustrates specifically the data required for a porosity determination and the method of calculation. Dry compressed air was used in the determination.

Data for the calculation of the volume of the bomb:

Barometric pressure (P_b) = 743.4 mm of mercury (14.376 lb. per sq. in.).

Temperature of the air in the bomb (T_B) = 70.5°F. (530.5°F. abs.).

Pressure of air in bomb (P_B) = 114.376 lb. per sq. in. abs.

Volume of air measured in burette (V_b) = 91.32 milliliters.

Temperature of air measured in the burette (T_b) = 73.9°F. (533.9°F. abs.).

Data for the calculation of the volume of voids in bomb with the specimen in it:

Barometric pressure (P_b) = 743.4 mm of mercury (14.376 lb. per sq. in.).

Temperature of the air in the bomb (T_B) = 70.7° F. (530.7° F. abs.).

Pressure of air in bomb (P_B) = 214.376 lb. per sq. in. abs.

Volume of air measured in burette (V_b) = 69.20 milliliters.

Temperature of air measured in the burette (T_b) = 74.1° F. (534.1° F. abs.).

Data for the calculation of the bulk volume of the specimen:

Weight of mercury displaced from pycnometer = 133.74 g.

Temperature of mercury = 75° F.

Table 2 gives factors expressing the deviation of air from Boyle's law (y) at several pressures and temperatures.

TABLE 2.— Values of y for use in equation 61/

Temperature, °F.	Pressure, lb. per sq. in. abs.							
	65	75	85	100	115	140	165	215
65.....	1.0013	1.0015	1.0018	1.0021	1.0025	1.0031	1.0036	1.0047
70.....	1.0012	1.0014	1.0016	1.0020	1.0023	1.0028	1.0034	1.0044
75.....	1.0011	1.0013	1.0015	1.0018	1.0021	1.0026	1.0031	1.0040
80.....	1.0010	1.0012	1.0014	1.0017	1.0020	1.0024	1.0028	1.0037
85.....	1.0009	1.0011	1.0013	1.0015	1.0018	1.0022	1.0026	1.0033
90.....	1.0008	1.0010	1.0011	1.0014	1.0016	1.0020	1.0023	1.0030
95.....	1.0007	1.0009	1.0010	1.0012	1.0014	1.0018	1.0021	1.0027
100.....	1.0007	1.0008	1.0009	1.0011	1.0013	1.0016	1.0019	1.0024

1/ The values in this table are based on equations taken from the International Critical Tables, vol. 3, p. 10. Interpolation was necessary to obtain some of the desired values.

y at 114.376 lb. per sq. in. and 70° F. = 1.0023 and
 at 214.376 " " " " " " = 1.0044.

Substituting in equation 6 the values obtained in the calibration of the bomb,

$$V_B = \frac{530.5 \times 91.32 \times 14.376}{533.9 [(1.0023 \times 114.376) - 14.376]} = 13.010 \text{ ml},$$

which is the volume of the empty bomb.

Then substituting in the same equation the values obtained with the test specimen in the bomb,

$$V_B = \frac{530.7 \times 69.20 \times 14.376}{534.1 [(1.0044 \times 214.376) - 14.376]} = 4.919 \text{ ml},$$

which is the volume of space occupied by compressed air in the bomb with the specimen in it. Therefore, the volume of the sand grains is the difference between 13.010 and 4.919, or 8.091 milliliters. The bulk volume of the specimen is equal to the volume of mercury displaced from the pycnometer times the specific volume of mercury at the temperature of the test (see table 1) or,

$$133.74 \times 0.073874 \text{ ml} = 9.88 \text{ ml}.$$

Therefore, the volume of pore space in the sample is,

$$9.88 - 8.091 = 1.789 \text{ ml}.$$

The volume of the pore space divided by the bulk volume multiplied by 100 equals the porosity in percent, or

$$\frac{1.789}{9.88} \times 100 = 18.11 \text{ percent porosity}.$$

In order to reduce the calculations to a minimum, two tables have been prepared for use in connection with equation 6. From table 3, a multiplier may be obtained to give the value of the ratio $\left(\frac{T_B}{T_b}\right)$ in the equation.

The proper multiplier is obtained as follows: (1) Subtract the temperature of the burette from that of the pressure bomb (values of these temperature differences are given in the first column of the table); (2) follow the indicated horizontal line to the column for the observed bomb temperature.

TABLE 3.- Values of $\left(\frac{T_B}{T_h}\right)$ for use in equation 6

Bomb temperature - burette temperature °F.	Bomb temperature, °F.							
	65	70	75	80	85	90	95	100
+5.0.....	1.0096	1.0095	1.0094	1.0093	1.0093	1.0092	1.0091	1.0090
+4.8.....	.0092	.0091	.0091	.0090	.0089	.0088	.0087	.0086
+4.6.....	.0088	.0088	.0087	.0086	.0085	.0084	.0084	.0083
+4.4.....	.0085	.0084	.0083	.0082	.0081	.0081	.0080	.0079
+4.2.....	.0081	.0080	.0079	.0078	.0078	.0077	.0076	.0076
+4.0.....	.0077	.0076	.0075	.0075	.0074	.0073	.0073	.0072
+3.8.....	.0073	.0072	.0072	.0071	.0070	.0070	.0069	.0068
+3.6.....	.0069	.0068	.0068	.0067	.0066	.0066	.0065	.0065
+3.4.....	.0065	.0065	.0065	.0063	.0063	.0062	.0062	.0061
+3.2.....	.0061	.0061	.0061	.0060	.0059	.0059	.0058	.0057
+3.0.....	.0057	.0057	.0057	.0056	.0055	.0055	.0054	.0054
+2.8.....	.0054	.0053	.0053	.0052	.0052	.0051	.0051	.0050
+2.6.....	.0050	.0049	.0049	.0048	.0048	.0047	.0047	.0047
+2.4.....	.0046	.0045	.0045	.0045	.0044	.0044	.0043	.0043
+2.2.....	.0042	.0042	.0042	.0041	.0041	.0040	.0040	.0039
+2.0.....	.0038	.0038	.0038	.0037	.0037	.0036	.0036	.0036
+1.8.....	.0034	.0034	.0034	.0033	.0033	.0033	.0033	.0032
+1.6.....	.0031	.0030	.0030	.0030	.0029	.0029	.0029	.0029
+1.4.....	.0027	.0026	.0026	.0026	.0026	.0026	.0025	.0025
+1.2.....	.0023	.0023	.0022	.0022	.0022	.0022	.0022	.0021
+1.0.....	.0019	.0019	.0019	.0019	.0018	.0018	.0018	.0018
+0.8.....	.0015	.0015	.0015	.0015	.0015	.0015	.0015	.0015
+0.6.....	.0011	.0011	.0011	.0011	.0011	.0011	.0011	.0011
+0.4.....	.0008	.0008	.0007	.0007	.0007	.0007	.0007	.0007
+0.2.....	.0004	.0004	.0004	.0004	.0004	.0004	.0004	.0004
-0.2.....	.9996	.9996	.9996	.9996	.9996	.9996	.9996	.9996
-0.4.....	.9992	.9992	.9993	.9993	.9993	.9993	.9993	.9993
-0.6.....	.9989	.9989	.9989	.9989	.9989	.9989	.9989	.9989
-0.8.....	.9985	.9985	.9985	.9985	.9985	.9985	.9986	.9986
-1.0.....	.9981	.9981	.9981	.9982	.9982	.9982	.9982	.9982
-1.2.....	.9977	.9977	.9978	.9978	.9978	.9978	.9978	.9979
-1.4.....	.9973	.9974	.9974	.9974	.9974	.9975	.9975	.9975
-1.6.....	.9970	.9970	.9970	.9970	.9971	.9971	.9971	.9972
-1.8.....	.9966	.9966	.9966	.9967	.9967	.9967	.9968	.9968
-2.0.....	.9962	.9962	.9963	.9963	.9963	.9964	.9964	.9964
-2.2.....	.9958	.9959	.9959	.9959	.9960	.9960	.9961	.9961
-2.4.....	.9954	.9955	.9955	.9956	.9956	.9957	.9957	.9957
-2.6.....	.9951	.9951	.9952	.9952	.9953	.9953	.9953	.9954
-2.8.....	.9947	.9947	.9948	.9948	.9949	.9949	.9950	.9950
-3.0.....	.9943	.9944	.9944	.9945	.9945	.9946	.9946	.9947
-3.2.....	.9939	.9940	.9941	.9941	.9942	.9942	.9943	.9943
-3.4.....	.9936	.9936	.9937	.9937	.9938	.9939	.9939	.9940
-3.6.....	.9932	.9933	.9933	.9934	.9934	.9935	.9936	.9936
-3.8.....	.9928	.9929	.9929	.9930	.9931	.9931	.9932	.9933
-4.0.....	.9924	.9925	.9926	.9926	.9927	.9928	.9928	.9929
-4.2.....	.9921	.9921	.9922	.9923	.9924	.9924	.9925	.9926
-4.4.....	.9917	.9918	.9918	.9919	.9920	.9921	.9921	.9922
-4.6.....	.9913	.9914	.9915	.9916	.9916	.9917	.9918	.9919
-4.8.....	.9909	.9910	.9911	.9912	.9913	.9913	.9914	.9915
-5.0.....	.9906	.9907	.9907	.9908	.9909	.9910	.9911	.9912

Values of the quantity, $\frac{P_b}{yP_B - P_b}$ when air is used may be ob-

tained from table 4 for use in equation 6. These values are suitable for the range of barometric pressures that pertain in the laboratory at Bartlesville, Okla., in which this work was done. The values of y used in computing this table are based on a temperature of 75°F. If a large number of determinations are to be made by this method at another altitude or at a temperature greatly different from this base temperature, a similar table covering the proper range of barometric pressures and temperatures should be prepared. Obviously, if table 4 or similar tables are to be used to simplify the calculation of test data, it is necessary to use only the bomb pressures listed in the table for the tests. The proper multiplier is found, first, by selecting the barometric pressure at which the test was made in the first column, and then following this horizontal line across the page to the column headed by the bomb pressure used in the test. By using tables 3 and 4, equation 6 becomes

$$V_B = (\text{a multiplier from table 3}) \times V_b \times (\text{a multiplier from table 4}),$$

or, in the example shown above, the volume of the bomb is calculated as follows:

$$V_B = 0.9936 \times 91.32 \times 0.14341 = 13.012 \text{ ml.}$$

TABLE 4.- Values of $\frac{P_b}{yP_B - P_b}$ for use in equation 6

Barometric pressure, mm of mercury	Bomb pressure				Lb. per sq. in., gage			
	50	60	70	85	100	125	150	200
725.0.....	0.28002	0.23330	0.19993	0.16460	0.13987	0.11184	0.09315	0.06980
725.2.....	.28009	.23336	.19999	.16464	.13990	.11187	.09318	.06982
725.4.....	.28017	.23343	.20004	.16469	.13994	.11190	.09321	.06984
725.6.....	.28025	.23349	.20010	.16474	.13998	.11193	.09323	.06986
725.8.....	.28033	.23356	.20015	.16478	.14002	.11196	.09326	.06988
726.0.....	.28040	.23362	.20021	.16483	.14006	.11199	.09328	.06990
726.2.....	.28048	.23369	.20026	.16487	.14010	.11202	.09331	.06992
726.4.....	.28056	.23375	.20032	.16492	.14014	.11205	.09333	.06994
726.6.....	.28064	.23381	.20037	.16496	.14017	.11208	.09336	.06996
726.8.....	.28071	.23388	.20043	.16501	.14021	.11211	.09339	.06997
727.0.....	.28079	.23394	.20048	.16505	.14025	.11215	.09341	.06999
727.2.....	.28087	.23401	.20054	.16510	.14029	.11218	.09344	.07001
727.4.....	.28095	.23407	.20059	.16514	.14033	.11221	.09346	.07003
727.6.....	.28102	.23414	.20065	.16519	.14037	.11224	.09349	.07005
727.8.....	.28110	.23420	.20070	.16523	.14041	.11227	.09351	.07007
728.0.....	.28118	.23427	.20076	.16528	.14044	.11230	.09354	.07009
728.2.....	.28125	.23433	.20081	.16532	.14048	.11233	.09356	.07011
728.4.....	.28133	.23439	.20087	.16537	.14052	.11236	.09359	.07013
728.6.....	.28141	.23446	.20093	.16542	.14056	.11239	.09362	.07015
728.8.....	.28148	.23452	.20098	.16546	.14060	.11242	.09364	.07017
729.0.....	.28156	.23459	.20104	.16551	.14064	.11245	.09367	.07019
729.2.....	.28164	.23465	.20109	.16555	.14068	.11248	.09369	.07021
729.4.....	.28172	.23471	.20114	.16560	.14071	.11252	.09372	.07022
729.6.....	.28179	.23478	.20120	.16564	.14075	.11255	.09374	.07024
729.8.....	.28187	.23484	.20126	.16569	.14079	.11258	.09377	.07026
730.0.....	.28195	.23491	.20131	.16573	.14083	.11261	.09380	.07028
730.2.....	.28203	.23497	.20137	.16578	.14087	.11264	.09382	.07030
730.4.....	.28211	.23504	.20142	.16583	.14091	.11267	.09385	.07032
730.6.....	.28217	.23519	.20147	.16586	.14094	.11270	.09387	.07034
730.8.....	.28225	.23517	.20152	.16591	.14098	.11273	.09390	.07036
731.0.....	.28233	.23522	.20158	.16596	.14102	.11276	.09392	.07038
731.2.....	.28241	.23529	.20164	.16601	.14106	.11279	.09395	.07040
731.4.....	.28249	.23535	.20169	.16605	.14110	.11283	.09398	.07042
731.6.....	.28257	.23542	.20175	.16610	.14114	.11286	.09400	.07044
731.8.....	.28265	.23549	.20181	.16615	.14118	.11289	.09403	.07046
732.0.....	.28273	.23555	.20186	.16619	.14122	.11292	.09406	.07048
732.2.....	.28279	.23560	.20191	.16623	.14125	.11295	.09408	.07049
732.4.....	.28287	.23567	.20196	.16628	.14129	.11298	.09410	.07051
732.6.....	.28295	.23574	.20202	.16632	.14133	.11301	.09413	.07053
732.8.....	.28303	.23580	.20208	.16637	.14137	.11304	.09415	.07055
733.0.....	.28311	.23587	.20214	.16642	.14141	.11307	.09418	.07057
733.2.....	.28319	.23594	.20219	.16646	.14145	.11310	.09421	.07059
733.4.....	.28327	.23600	.20225	.16651	.14149	.11314	.09423	.07061

TABLE 4.— Values of $\frac{P_b}{\sqrt{P_B - P_b}}$ for use in equation 6 (Cont'd)

Barometric pressure, mm of mercury	Bomb pressure				Lb. per sq. in., gage			
	50	60	70	85	100	125	150	200
733.6.....	0.28333	0.23605	0.20229	0.16655	0.14152	0.11316	0.09425	0.07063
733.8.....	.28341	.23612	.20235	.16659	.14156	.11319	.09428	.07065
734.0.....	.28349	.23619	.20241	.16664	.14160	.11322	.09431	.07067
734.2.....	.28357	.23625	.20246	.16669	.14164	.11326	.09433	.07069
734.4.....	.28365	.23632	.20252	.16673	.14168	.11329	.09436	.07071
734.6.....	.28373	.23639	.20258	.16678	.14172	.11332	.09439	.07073
734.8.....	.28381	.23645	.20263	.16683	.14176	.11335	.09441	.07075
735.0.....	.28389	.23652	.20269	.16687	.14180	.11338	.09444	.07077
735.2.....	.28395	.23657	.20273	.16691	.14183	.11341	.09446	.07078
735.4.....	.28403	.23664	.20279	.16696	.14187	.11344	.09449	.07080
735.6.....	.28411	.23670	.20285	.16700	.14191	.11347	.09451	.07082
735.8.....	.28419	.23677	.20291	.16705	.14195	.11350	.09454	.07084
736.0.....	.28427	.23684	.20296	.16710	.14199	.11354	.09457	.07086
736.2.....	.28435	.23690	.20302	.16714	.14203	.11357	.09459	.07088
736.4.....	.28443	.23697	.20308	.16719	.14207	.11360	.09462	.07090
736.6.....	.28451	.23704	.20313	.16724	.14211	.11363	.09465	.07092
736.8.....	.28457	.23709	.20319	.16727	.14214	.11366	.09467	.07094
737.0.....	.28465	.23715	.20325	.16732	.14218	.11369	.09469	.07096
737.2.....	.28473	.23722	.20330	.16737	.14222	.11372	.09472	.07098
737.4.....	.28481	.23728	.20336	.16741	.14226	.11375	.09475	.07100
737.6.....	.28489	.23735	.20340	.16746	.14230	.11378	.09477	.07102
737.8.....	.28497	.23742	.20346	.16751	.14234	.11381	.09480	.07104
738.0.....	.28505	.23748	.20352	.16755	.14238	.11385	.09483	.07106
738.2.....	.28511	.23753	.20358	.16759	.14241	.11387	.09485	.07107
738.4.....	.28519	.23760	.20363	.16764	.14245	.11390	.09487	.07109
738.6.....	.28527	.23767	.20369	.16768	.14249	.11393	.09490	.07111
738.8.....	.28535	.23773	.20375	.16773	.14253	.11396	.09493	.07113
739.0.....	.28543	.23780	.20379	.16778	.14257	.11400	.09495	.07115
739.2.....	.28551	.23787	.20385	.16782	.14261	.11403	.09498	.07117
739.4.....	.28559	.23793	.20390	.16787	.14265	.11406	.09501	.07119
739.6.....	.28567	.23800	.20396	.16792	.14269	.11409	.09503	.07121
739.8.....	.28573	.23805	.20402	.16795	.14272	.11412	.09505	.07122
740.0.....	.28581	.23812	.20407	.16800	.14276	.11415	.09508	.07124
740.2.....	.28589	.23818	.20413	.16805	.14280	.11418	.09510	.07126
740.4.....	.28597	.23825	.20419	.16809	.14284	.11421	.09513	.07128
740.6.....	.28605	.23832	.20423	.16814	.14288	.11425	.09516	.07130
740.8.....	.28613	.23838	.20429	.16819	.14292	.11428	.09518	.07132
741.0.....	.28621	.23845	.20434	.16823	.14296	.11431	.09521	.07134
741.2.....	.28627	.23850	.20439	.16827	.14299	.11433	.09523	.07136
741.4.....	.28635	.23857	.20444	.16832	.14303	.11436	.09526	.07138
741.6.....	.28643	.23863	.20450	.16836	.14307	.11440	.09528	.07140
741.8.....	.28651	.23870	.20456	.16841	.14311	.11443	.09531	.07142
742.0.....	.28659	.23877	.20462	.16846	.14315	.11446	.09534	.07144
742.2.....	.28667	.23883	.20467	.16850	.14318	.11449	.09536	.07146
742.4.....	.28675	.23890	.20473	.16855	.14322	.11452	.09539	.07148
742.6.....	.28683	.23897	.20479	.16860	.14326	.11456	.09542	.07150
742.8.....	.28688	.23902	.20483	.16863	.14329	.11458	.09544	.07151

TABLE 4.— Values of $\frac{P_b}{yP_B - P_b}$ for use in equation 6 (Cont'd)

Barometric pressure, mm of mercury	Bomb pressure				Lb. per sq. in., gage			
	50	60	70	85	100	125	150	200
743.0.....	0.28696	0.23908	0.20489	0.16868	0.14333	0.11461	0.09546	0.07153
743.2.....	.28704	.23915	.20494	.16873	.14337	.11464	.09549	.07155
743.4.....	.28712	.23922	.20500	.16877	.14341	.11468	.09552	.07157
743.6.....	.28720	.23928	.20506	.16882	.14345	.11471	.09554	.07159
743.8.....	.28728	.23935	.20511	.16887	.14349	.11474	.09557	.07161
744.0.....	.28736	.23941	.20517	.16892	.14353	.11477	.09560	.07163
744.2.....	.28742	.23946	.20521	.16895	.14356	.11480	.09562	.07165
744.4.....	.28750	.23953	.20527	.16900	.14360	.11483	.09564	.07167
744.6.....	.28758	.23960	.20533	.16904	.14364	.11486	.09567	.07169
744.8.....	.28766	.23966	.20539	.16909	.14368	.11489	.09570	.07171
745.0.....	.28774	.23973	.20544	.16914	.14372	.11492	.09572	.07173
745.2.....	.28782	.23980	.20550	.16919	.14376	.11495	.09575	.07175
745.4.....	.28790	.23986	.20556	.16923	.14380	.11499	.09576	.07177
745.6.....	.28798	.23993	.20561	.16928	.14384	.11502	.09580	.07179
745.8.....	.28804	.23998	.20566	.16931	.14387	.11504	.09582	.07180
746.0.....	.28812	.24005	.20571	.16936	.14391	.11507	.09585	.07182
746.2.....	.28820	.24011	.20577	.16941	.14395	.11511	.09588	.07184
746.4.....	.28828	.24018	.20583	.16946	.14399	.11514	.09590	.07186
746.6.....	.28836	.24025	.20588	.16950	.14403	.11517	.09593	.07188
746.8.....	.28844	.24031	.20594	.16955	.14407	.11520	.09595	.07190
747.0.....	.28852	.24038	.20600	.16960	.14411	.11523	.09598	.07192
747.2.....	.28858	.24043	.20604	.16963	.14414	.11526	.09600	.07194
747.4.....	.28866	.24050	.20610	.16968	.14418	.11529	.09603	.07196
747.6.....	.28874	.24056	.20616	.16973	.14422	.11532	.09605	.07198
747.8.....	.28882	.24063	.20621	.16977	.14426	.11535	.09608	.07200
748.0.....	.28890	.24069	.20627	.16982	.14430	.11539	.09611	.07202
748.2.....	.28898	.24076	.20632	.16987	.14434	.11542	.09613	.07204
748.4.....	.28906	.24083	.20638	.16991	.14438	.11545	.09616	.07206
748.6.....	.28914	.24089	.20644	.16996	.14442	.11548	.09619	.07208
748.8.....	.28920	.24094	.20648	.17000	.14445	.11551	.09621	.07209
749.0.....	.28928	.24101	.20654	.17004	.14449	.11554	.09623	.07211
749.2.....	.28936	.24108	.20660	.17009	.14453	.11557	.09626	.07213
749.4.....	.28944	.24114	.20665	.17014	.14457	.11560	.09629	.07215
749.6.....	.28952	.24121	.20671	.17018	.14461	.11563	.09631	.07217
749.8.....	.28960	.24128	.20677	.17023	.14465	.11566	.09634	.07219
750.0.....	.28968	.24134	.20683	.17028	.14469	.11570	.09637	.07221
750.2.....	.28974	.24139	.20687	.17031	.14472	.11572	.09639	.07223
750.4.....	.28982	.24146	.20693	.17036	.14476	.11575	.09641	.07225
750.6.....	.28990	.24153	.20698	.17041	.14480	.11578	.09644	.07226
750.8.....	.28998	.24159	.20704	.17045	.14484	.11582	.09647	.07228
751.0.....	.29006	.24166	.20710	.17050	.14488	.11585	.09649	.07230
751.2.....	.29014	.24173	.20715	.17055	.14492	.11588	.09652	.07232
751.4.....	.29022	.24179	.20721	.17059	.14496	.11591	.09655	.07234
751.6.....	.29030	.24186	.20727	.17064	.14500	.11594	.09657	.07236
751.8.....	.29036	.24191	.20731	.17068	.14503	.11597	.09659	.07238
752.0.....	.29044	.24198	.20737	.17072	.14507	.11600	.09662	.07240

TABLE 4.— Values of $\frac{P_b}{yP_B - P_b}$ for use in equation 6 (Cont'd)

Barometric pressure, mm of mercury	Bomb pressure				Lb. per sq. in., gage			
	50	60	70	85	100	125	150	200
752.2.....	0.29052	0.24204	0.20742	0.17077	0.14511	0.11603	0.09665	0.07242
752.4.....	.29060	.24211	.20748	.17082	.14515	.11606	.09667	.07244
752.6.....	.29068	.24218	.20754	.17086	.14519	.11610	.09670	.07246
752.8.....	.29076	.24224	.20760	.17091	.14523	.11613	.09673	.07248
753.0.....	.29084	.24231	.20765	.17096	.14527	.11616	.09675	.07250
753.2.....	.29092	.24238	.20771	.17100	.14531	.11619	.09678	.07252
753.4.....	.29098	.24243	.20775	.17104	.14534	.11621	.09680	.07253
753.6.....	.29106	.24249	.20781	.17109	.14538	.11625	.09683	.07255
753.8.....	.29114	.24256	.20787	.17113	.14542	.11628	.09685	.07257
754.0.....	.29122	.24262	.20792	.17118	.14546	.11631	.09688	.07259
754.2.....	.29130	.24269	.20798	.17123	.14550	.11634	.09690	.07261
754.4.....	.29138	.24276	.20804	.17127	.14554	.11637	.09693	.07263
754.6.....	.29146	.24282	.20809	.17132	.14558	.11641	.09696	.07265
754.8.....	.29152	.24287	.20814	.17136	.14561	.11643	.09698	.07267
755.0.....	.29160	.24294	.20819	.17140	.14565	.11646	.09700	.07269
755.2.....	.29168	.24301	.20825	.17145	.14569	.11649	.09703	.07271
755.4.....	.29176	.24307	.20831	.17150	.14573	.11653	.09706	.07273
755.6.....	.29184	.24314	.20837	.17154	.14577	.11656	.09708	.07275
755.8.....	.29192	.24321	.20842	.17159	.14581	.11659	.09711	.07277
756.0.....	.29200	.24327	.20848	.17164	.14585	.11662	.09714	.07279
756.2.....	.29208	.24334	.20854	.17169	.14589	.11665	.09716	.07281
756.4.....	.29214	.24339	.20858	.17172	.14592	.11668	.09718	.07282
756.6.....	.29222	.24346	.20864	.17177	.14596	.11671	.09721	.07284
756.8.....	.29230	.24352	.20869	.17181	.14600	.11674	.09724	.07286
757.0.....	.29238	.24359	.20875	.17186	.14604	.11677	.09726	.07288
757.2.....	.29246	.24366	.20881	.17191	.14608	.11680	.09729	.07290
757.4.....	.29254	.24372	.20886	.17196	.14612	.11684	.09732	.07292
757.6.....	.29262	.24379	.20892	.17200	.14616	.11687	.09734	.07294
757.8.....	.29268	.24384	.20896	.17204	.14619	.11689	.09736	.07296
758.0.....	.29276	.24391	.20902	.17208	.14623	.11692	.09739	.07298
758.2.....	.29284	.24397	.20908	.17213	.14627	.11696	.09742	.07300
758.4.....	.29292	.24404	.20913	.17218	.14631	.11699	.09744	.07302
758.6.....	.29300	.24410	.20919	.17223	.14635	.11702	.09747	.07304
758.8.....	.29307	.24417	.20925	.17227	.14639	.11705	.09750	.07306
759.0.....	.29315	.24424	.20931	.17232	.14643	.11708	.09752	.07308
759.2.....	.29323	.24430	.20936	.17237	.14647	.11712	.09755	.07310
759.4.....	.28329	.24435	.20941	.17240	.14650	.11714	.09757	.07311
759.6.....	.28337	.24442	.20946	.17245	.14654	.11717	.09760	.07313
759.8.....	.28345	.24449	.20952	.17250	.14658	.11720	.09762	.07315
760.0.....	.28353	.24455	.20958	.17254	.14662	.11724	.09765	.07317

Similarly, the volume of voids in the bomb with the test specimen in it is:

$$V_B = 0.9936 \times 69.20 \times 0.07157 = 4.921 \text{ ml.}$$

Accuracy of the Method

The sources of error in the determination of porosity by this method comprise those in measuring pressure, volume, and temperature. The effect of a given error in any of these measurements will depend on the particular conditions of the test. To illustrate the effect of an error in a pressure measurement, the preceding example may be considered further. If, in this example, the actual pressure on the bomb, with the specimen in it, was 200.2 pounds per square inch, gage, instead of 200.0 pounds, as read, the volume of the pores in the test specimen would be 1.784 milliliters instead of 1.789 milliliters, as previously calculated. The porosity then becomes 18.06 percent instead of 18.11 percent.

The porosities of several specimens have been determined by means of a range of pressures from 40 to 250 pounds per square inch in order to study the effect of pressure. In every case, the results checked very closely, indicating that neither the bomb nor the test specimen was affected by the pressures used. Table 5 gives the results of these tests on two specimens.

The use of disks, collars, and slugs in the bomb to reduce the volume of space in it unoccupied by the specimen results in a greater accuracy, which is due in part to the fact that a higher pressure may be used in the bomb without exceeding the capacity of the burette when the volume of the gas is measured at atmospheric pressure. The dead-weight gage is more sensitive at higher pressures, and, also, an error of one or two tenths pound will mean a smaller percentage error in the pressure measurement. If, in the example considered, a pressure of 50 pounds per square inch, gage, had been used and an error of 0.2 pound had been made in the pressure measurement (50.2), the determination of the void space in the bomb would be in error by approximately 0.4 percent or by 0.020 milliliters as compared with 0.005 milliliters when 200 pounds was used, with an assumed error of 0.2 pound. With this error in the determination made at 50 pounds per square inch pressure, the porosity in the above example would be 17.91 rather than 18.11 percent. Furthermore, it is desirable to reduce the volume of unoccupied space in the pressure bomb, due to the fact that the volume of the sand grains in the sample is not measured directly but is equal to the difference between the volume of the bomb and the volume of void space remaining in the bomb when the core is placed in it. The percentage error in either of these two volumes is greatly magnified in their difference, and the magnification will vary directly as the two volumes. In other words, if the volume of the void space remaining in the bomb when a specimen is placed in it is twice as great as when the specimen is placed in another bomb, a given percentage error in the determination of the volume of void space remaining in the bombs would result in twice as great an error in the porosity.

TABLE 5.-- A comparison of porosity determinations
made at different pressures

Pressure, pounds per square inch, gage	Volume of voids, ml	Volume of sand grains, ml	Porosity, percent	Remarks
Core No. 421				
50	10.650	11.516	24.04	Volume of bomb = 22.166 ml. Bulk volume of specimen = 15.16 ml.
75	10.651	11.515	24.04	
100	10.647	11.519	24.02	
120	10.657	11.509	24.08	
130	10.659	11.507	24.10	
Core No. 490				
40	5.806	8.205	17.95	Volume of bomb = 14.011 ml. Bulk volume of specimen = 10.00 ml.
60	5.819	8.192	18.08	
80	5.813	8.198	18.02	
100	5.814	8.197	18.03	
120	5.817	8.194	18.06	
150	5.816	8.195	18.05	
180	5.821	8.190	18.10	
210	5.815	8.196	18.04	

When a burette of good grade is used and care is taken in the readings, the error in any reading should not exceed ± 0.02 milliliters. If the error in reading the burette in the above example, before and after the gas was expanded into it, had been cumulative, the correct volume of atmospheric gas might have been as much as 69.24 instead of 69.20 milliliters. This would give 4.922 milliliters as the volume of void space, or 1.792 milliliters as the volume of the pores, instead of 1.789 milliliters, as previously calculated. The porosity then becomes 18.14 percent instead of 18.11 percent. The effect of a given error in the volume, as measured in the burette, depends upon the magnitude of the volume measured and the magnitude of the pressure used. If air is expanded from a pressure of ten atmospheres, a given error in reading the volume will affect the porosity value approximately half as much as if it had been expanded from only 5 atmospheres.

With good thermometers, the temperature should be measured easily to within $\pm 0.2^\circ\text{F}$. If, instead of the temperatures observed in the example, the actual bomb temperature was 70.9°F . and the burette temperature 73.9°F ., the volume of voids in the bomb would be 4.923 milliliters, which, in turn, would give a pore space of 1.793 milliliters instead of 1.789 milliliters and a porosity of 18.15 percent instead of 18.11 percent.

Certain precautions should be observed in the determination of the volume of the sand grains in order to eliminate possible errors due to the manipulation of the apparatus. Before an initial test with the apparatus is made, stopcock E (fig. 1) should be opened to the burette and stopcock D turned to open the burette to the atmosphere, so as to bring the manometer into balance. This should not be necessary on subsequent tests, as the monometer is balanced at the conclusion of each test. If the room temperature is high, the vapor pressure of the water in the manometer may throw it out of balance during the course of a test. This may be avoided by replacing the water with a fluid having a low vapor pressure. Dibutylphthalate has been found satisfactory for this purpose, as its density is low and its vapor pressure is even less than that of mercury. Leaks are an ever present source of error, therefore all fittings and joints should be tested for leaks periodically with a soap solution.

Washburn and Bunting^{9/} pointed out that with certain ceramic bodies, some gases are adsorbed^{10/} and therefore higher porosities are indicated by gas expansion methods than actually exist. Dry air, although adsorbed to a certain extent by some materials, has been used, as it is the most convenient gas for the determination of porosity in the average laboratory. To determine the magnitude of the error due to adsorption where air is used, a number of porosity measurements made with air have been duplicated with helium. The adsorption of this gas by most materials at room temperature is negligible.

9/ See footnote 6.

10/ International Critical Tables, vol. 3, pp. 250-51.

Table 6 gives the results obtained on 12 samples with both gases. For each sample, the porosity obtained with air is larger, but the difference in the two values is not excessive. Therefore, it is believed that dry air should be satisfactory for most work. However, if extremely accurate results are desired, helium or hydrogen should be used.

The greatest inaccuracy in the method is in the determination of the bulk volume of the specimen. Check determinations have been made on more than 200 test specimens, and on about 90 percent of these the results of the bulk volume determination checked within 0.2 percent, or within 0.02 ml on a 10-milliliter specimen. If in the above example the bulk volume of the test specimen had been found to be 9.90 ml instead of 9.88 ml, the porosity would be 18.27 instead of 18.11 percent. If the test specimen is allowed to touch the lid before all the air is expelled from the pycnometer, a bubble of air will remain in the pycnometer and cause an error of 0.03 or 0.04 ml. As explained, this error can be avoided by holding the specimen below the surface with a stiff steel wire. The pycnometer should be handled as little as possible so as to avoid any appreciable change in the temperature of the mercury during a determination. With reasonable care the temperature of the mercury will not change more than 2° F. during a determination, even under unfavorable weather conditions. This will mean a change of only 0.02 percent in the specific volume of mercury.

The magnitude of the total probable error in a porosity determination will depend on the size of the specimen, but, as the several possible errors are compensative, their overall effect will be less than their sum. It is believed that where a test specimen is used having a displacement of 8 or 10 milliliters, as in the example used in this report, the total probable error will not exceed ± 0.25 percent in the porosity value. The true porosity of the specimen used in the example, then, is equal to 18.11 ± 0.25 percent, or some value between 17.86 and 18.36 percent. If a specimen having a displacement of 3 or 4 milliliters is used, the total error probably would be twice as great; and, conversely, if a larger specimen is used, the probable error will be less.

Advantages of the Method

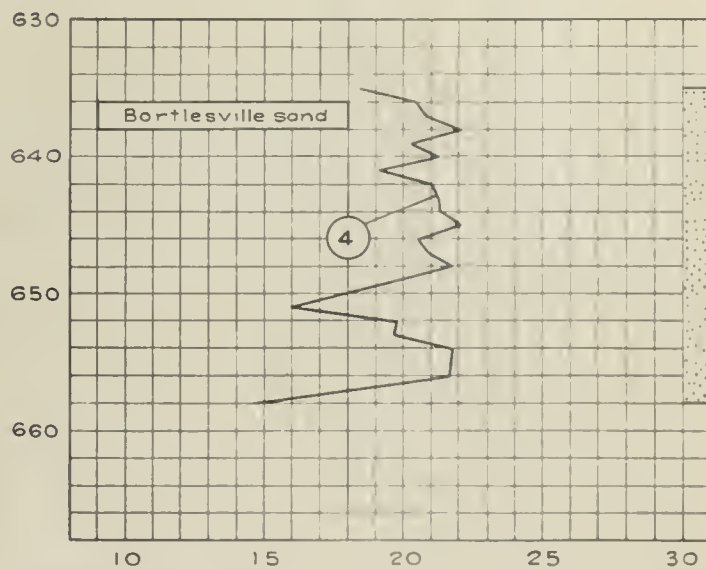
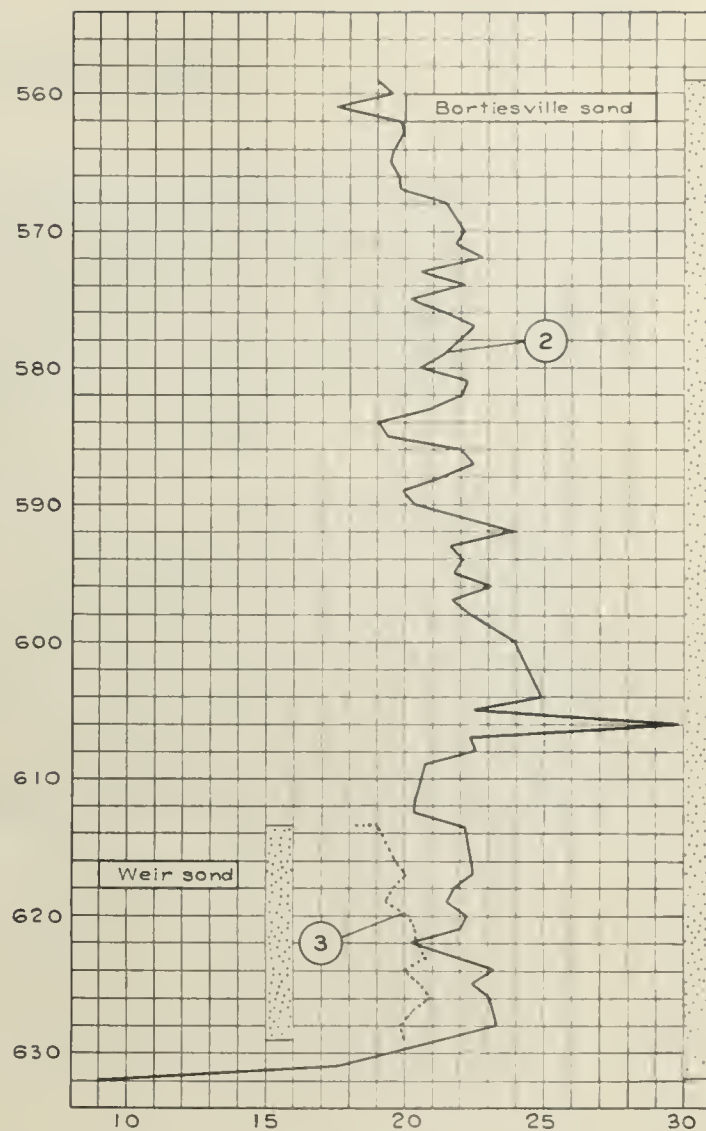
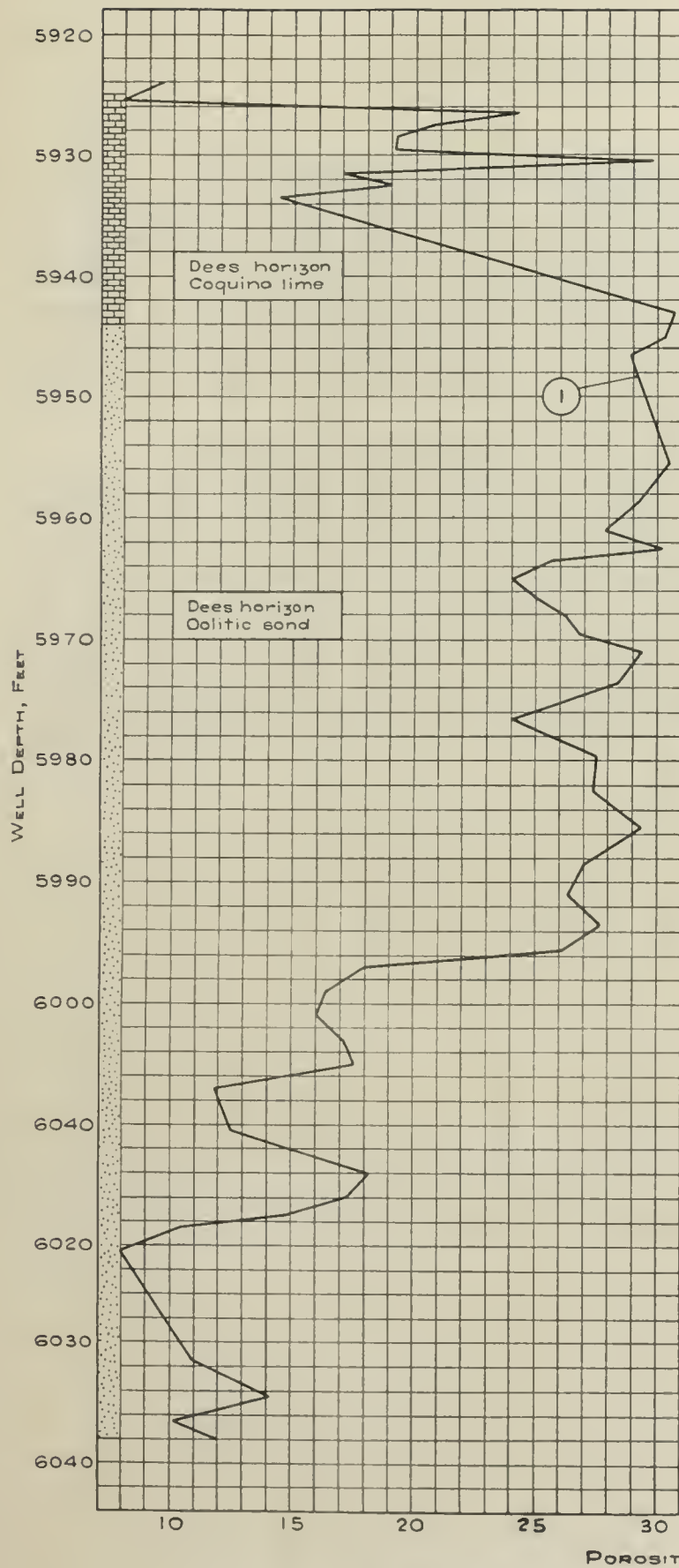
Probably the greatest advantage of the method is that the test specimen is not affected in any way by the determination and is available for other tests; or, if it is desired, check porosity determinations may be made. Very accurate results may be obtained, although the time required for a determination is not excessive. If the volume of the bomb is known from previous tests, the determination of the volume of the sand grains can be made in from 5 to 10 minutes. The determination of the bulk volume of the core by the displacement of mercury will require about the same length of time. When a number of cores are tested, 15 minutes should be sufficient for each.

TABLE 6.-- Comparison of porosity values obtained using air and helium

Core no.	Sand	Bulk volume of specimen, ml	Volume of voids, ml		Volume of sand grains, ml		Porosity, percent		Percent difference
			With air	With helium	With air	With helium	With air	With helium	
309.....	Bartlesville	10.12	6.567	6.536	7.932	7.963	21.62	21.31	0.31
374.....	do.	10.395	6.327	6.291	8.172	8.208	21.39	21.04	0.35
381.....	do.	9.58	7.034	7.005	7.465	7.494	22.08	21.77	0.31
390.....	do.	10.145	6.618	6.596	7.881	7.903	22.32	22.10	0.22
392.....	do.	9.89	6.659	6.615	7.840	7.888	20.73	20.24	0.49
411.....	do.	11.15	5.312	5.278	9.187	9.221	17.61	17.30	0.31
429.....	Oolitic sandstone, Dees horizon	8.105	7.559	7.550	6.940	6.949	14.37	14.26	0.16
452.....	do.	9.635	6.867	6.843	7.632	7.656	20.79	20.54	0.25
479.....	do.	10.42	5.786	5.770	8.713	8.729	16.38	16.23	0.15
528.....	Weir	10.02	6.442	6.409	8.057	8.090	19.59	19.26	0.33
529.....	do.	10.94	5.673	5.635	8.826	8.864	19.32	18.98	0.34
530.....	do.	9.425	6.975	6.948	7.524	7.551	20.17	19.88	0.29

1/ Volume of bomb = 14.499 ml.

Each core was subjected to both gases under the same pressure.



- 1 Rodesso field, Sec. 23, T. 23 N., R. 16 W., Louisiana
- 2 Delaware-Childers field, Sec. 2, T. 26 N., R. 16 E., Oklahoma
- 3 Johnson-Magoffin field, Johnson County, Kentucky
- 4 Delaware-Childers field, Sec. 36, T. 27 N., R. 16 E., Oklahoma

Figure 2.—Porosity-depth profile of four producing wells.

LIST OF POROSITIES OF OIL SANDS

The method of determining porosities of oil-sand specimens described in the forward section of this report has been used extensively in conjunction with research that pertains to subsurface conditions in various oil and gas fields. In the course of the studies by Bureau of Mines engineers, several hundred porosity determinations have been made, and the results of these tests on cores from wells in 11 States have been recorded in table 7 of this report in order that the data may be generally available.

All of the porosity values that have been tabulated should not be thought to represent the average porosity of the formation from which the samples were taken. For many formations, only a few pieces of core were available, and the porosities measured may or may not be representative of the entire formation. In figure 2, the porosity of the sand in four producing wells is plotted against the respective well depths in order to show the manner in which the porosity varies in a producing formation. At several points the porosity changes as much as 5 to 10 percent within 1 foot of sand. This is especially true in the profile shown for the sands in the Rodessa (La.) field.

TABLE 7.— Porosities of oil sands

Field	State	Sand	Location	Number of cores tested	Range of porosity	Average porosity, percent	Remarks
Garland City.....	Arkansas	Upper Trinity	Sec. 3-T16S-R26W	2	28.7-29.9	29.3	
Iles Dome.....	Colorado	Sundance		1	16.8	16.8	
	Illinois	Kimmiswich L.S.	Dupo, Ill.	4	11.6-16.5	15.8	
	do.	Makanda S.S.	Bosky Dell, Ill.	4	16.2-19.6	18.3	Outcrop.
Barton Co.	Kansas	Lansing-Kansas City L.S.	Sec. 16-T20S-R11W	5	6.2-25.3	12.6	
	do.	Basal	Sec. 12-T15S-R20W	2	12.5-16.3	14.4	
Chautauqua Co. ...	do.	Peru	Sec. 22-T34S-R11E	2	21.6-23.2	22.4	
Gates.....	do.	Arbuckle L.S.	Sec. 26-T21S-R13W	3	14.0-17.5	15.5	
Hollow Pool.....	do.	Hunton L.S.	Sec. 20-T22S-R3W	1	23.6	23.6	
Rice County.....	do.	Siliceous L.S.	Sec. 21-T20S-R10W	1	17.2	17.2	
	do.	do.	Sec. 31-T22S-R11W	1	12.2	12.2	
Johnson-Magoffin..	Kentucky	Weir		15	19.0-20.9	20.0	
Bosco.....	Louisiana	Hernandez	Sec. 35-T8S-R3E	2	29.8-30.8	30.3	
Caddo.....	do.	Upper Tokio		1	36.3	35.3	
Cleiborne Parish..	do.	Nacatoch	Sec. 2-T22N-R7W	2	31.8-33.8	32.8	
Cotton Valley.....	do.		Sec. 28-T21N-R10W	1	24.8	24.8	
Bosco.....	do.	Mire	Sec. 34-T8S-R3E	1	26.2	26.2	
Do.	do.	2nd Marginnelina		1	31.2	31.2	
	do.	Tokio	Sec. 2-T22N-R7W	2	29.3-30.7	30.0	
Homer.....	do.	Limestone	Sec. 24-T21N-R8W	2	18.9-21.7	20.3	
Do.	do.	Sandstone	Sec. 24-T21N-R8W	4	28.9-36.3	32.0	
Rodessa.....	do.	Hill horizon; sandstone.	Sec. 21-T23N-R16W	2	18.0-22.8	20.4	
Do.	do.	Dees horizon; sandstone.	Sec. 21-T23N-R16W	3	13.1-24.9	17.9	
Do.	do.	Dees horizon; sandstone.	Sec. 21-T23N-R16W	7	22.2-28.1	24.4	
Do.	do.	Coquina limestone.	Sec. 21-T23N-R16W	5	12.5-24.2	19.4	
Do.	do.	Dees horizon; oolitic sandstone.					
Do.	do.	Dees horizon; oolitic sandstone.	Sec. 23-T23N-R16W	13	7.0-29.7	17.2	
Do.	do.	Coquina limestone.					
Do.	do.	Dees horizon; oolitic sandstone.	Sec. 23-T23N-R16W	69	8.0-32.0	22.2	

TABLE 7.-- Porosities of oil sands (Cont'd)

Field	State	Sand	Location	Number of cores tested	Range of porosity	Average porosity, percent	Remarks
Cattaraugus County.	Louisiana	Top of Trinity	Sec. 13-T21N-R5W	2	20.3-26.4	23.4	
Adair.....	New York	Chipmunk	Lot 24, Elko Twp-R8	2	14.5-15.1	14.8	
Alluwe.....	Oklahoma	Bartlesville	Sec. 1-T26N-R15E	1	26.2	26.2	
Bartlesville.....	do.	do.	Sec. 8-T25N-R17E	22	8.7-21.8	18.7	
Do.	do.	Wayside	Sec. 5-T25N-R13E	4	15.7-19.4	18.0	
Burbank.....	do.	Peru	Sec. 5-T25N-R13E	17	14.7-20.2	18.6	
Coody's Bluff.....	do.	Burbank	Sec. 18-T26N-R17E	2	15.2-15.7	15.4	
Del-Childers.....	do.	Bartlesville	Sec. 2-T26N-R16E	3	16.6-24.5	21.6	
Do.	do.	do.	Sec. 36-T27N-R16E	71	8.6-25.0	21.0	
Fitts Pool.....	do.	First Bromide Sd.	Sec. 20-T18N-R4W	27	14.6-25.8	20.7	
Logan County.....	do.	Second Simpson Sd.	Sec. 3-T11N-R3W	1	7.7	7.7	
Oklahoma City.....	do.	Wilcox	Sec. 11-T11N-R3W	2	14.9-15.1	15.0	
Do.	do.	Mollman	Sec. 13-T11N-R3W	10	16.9-23.3	20.7	
Do.	do.	Hammer-Haindl	Sec. 14-T11N-R3W	2	12.5-15.7	14.1	
Do.	do.	do.	Sec. 25-T11N-R3W	3	7.1-13.1	10.9	
Do.	do.	Johnson	Sec. 30-T12N-R3W	8	10.5-24.0	15.1	
Do.	do.	Hammer-Haindl	Sec. 30-T12N-R3W	3	18.2-26.6	22.3	
Do.	do.	Johnson	Sec. 30-T12N-R3W	1	26.5	26.5	
Do.	do.	Wilcox	Sec. 30-T12N-R3W	2	24.3-30.3	27.3	
Sasakwa.....	do.	Booch	Sec. 31-T11N-R2W	2	19.4-21.8	20.6	
West Chandler.....	do.	Skinner	Sec. 16-T7N-R8E	1	17.5	17.5	
W. Holdenville.....	do.	Booch	Howard County,	1	12.1	12.1	
Wetumka Area.....	do.	Cromwell	Sec. 13-Blk. 32-T2S	4	22.0-25.4	23.2	
Bradford.....	Pennsylvania	Bradford	Gregg County,	1	12.1	12.1	
Big Springs.....	Texas	Permian L.S.	W. P. Chisom survey	5	13.1	13.1	
East Texas.....	do.	Woodbine	D. Ferguson survey	8	11.9-22.2	17.9	
Do.	do.	do.	Gregg County,	1	23.8-30.1	27.3	
Do.	do.	do.	B. N. Hampton survey	3	26.8	26.8	
Do.	do.	do.		2	21.8-25.1	23.5	
				3	28.0-29.7	28.9	

TABLE 7.- Porosities of oil sands (Cont'd)

Field	State	Sand	Location	Number of cores tested	Range of porosity	Average porosity, percent	Remarks
East Texas (Cont'd)	Texas	Woodbine	Gregg County, J. Wilkinson survey	1	23.8	23.8	
Do.	do.	do.	Gregg County, J. Mosley survey	2	27.5-29.4	28.5	
Do.	do.	do.	Gregg County, F. W. Johnson survey	4	19.8-31.4	26.7	
Do.	do.	do.	Gregg County, G. A. Thompson survey	1	24.0	24.0	
Do.	do.	do.	Gregg County, P. Warrenner survey	1	27.9	27.9	
Big Medicine Bow. Salt Creek.....	Wyoming do.	Second Bench-Sundance Second Wall Creek	Sec. 26-T21N-R79W	1 2	15.0 18.1-19.9	15.0 19.0	

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REPORT OF INVESTIGATIONS

EARTH VIBRATIONS CAUSED BY QUARRY BLASTING

PROGRESS REPORT I



BY

J. R. THOENEN AND STEPHEN L. WINDES

REPORT OF INVESTIGATIONS

UNITED STATES DEPARTMENT OF THE INTERIOR - BUREAU OF MINES

EARTH VIBRATIONS CAUSED BY QUARRY BLASTING^{1/}

PROGRESS REPORT I

By J. R. Thoenen^{2/} and Stephen L. Windes^{3/}

INTRODUCTION

This paper outlines briefly results of seismic measurements recorded in the vicinity of quarry blasting under practical operating conditions. The data cover field work done from January to the middle of August 1936, during which time records were made of 61 blasts in 19 quarries. Of the tests made, one was in biotite-gneiss, two in dolomite, two in a flintlike volcanic complex, three in gabbrodiorite, five in trap rock, and six in limestone.

One quarry was in Alabama, two were in South Carolina, two in New York, four in Kentucky, five in Connecticut, and five in Massachusetts. Explosive charges ranged from 0.36 pound to 41,800 pounds, and the method of shooting varied from mud caps through hammer and wagon drill holes to churn drill holes and coyote tunnel shots.

Early small-scale experiments had indicated that certain field technique should be followed to obtain accurate results^{4/}. However, as in most lines of research, additional information and experience gained in the field indicated that it was necessary to change field procedure radically. For this reason the period covered by this paper must be considered still formative, during which errors in operations and procedure were discovered and corrected as rapidly as possible. However, since a blast once shot cannot be duplicated exactly, and since several of the shots recorded would not have been repeated for several months, it happened that in some instances records were incomplete and much of their value was lost due to faulty technique. In addition, several minor mechanical difficulties were encountered in field operation of the equipment, which also caused errors in records.

^{1/} The Bureau of Mines will welcome reprinting of this paper provided the following footnote acknowledgment is used: "Reprinted from Bureau of Mines Report of Investigations 3353."

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^{4/} Lee, F. W., Thoenen, J. R., Windes, S. L., Earth Vibrations Caused by Quarry Blasting: Report of Investigations 3319, Bureau of Mines, 1936, p. 19.

To add to these difficulties, some records obtained, even though free from errors of technique or mechanical operation, were directly contradictory in relation to other but similar conditions. For these reasons, and because the variable factors are so numerous, the data contained in this paper are presented as a progress report, and all conclusions reached are tentative only.

Final interpretations and conclusions must await the results of more field tests that are being conducted under better-controlled conditions and with a better understanding of field operation.

DETAILS OF TESTS

Quarry A.

A single shot was made at this quarry with the charge placed in 17 well-drill holes 6 inches in diameter by an average depth of 130 feet. The holes were drilled in dolomite, the beds ranging from 5 to 30 feet in thickness and having some thin clay partings. The beds dipped into the quarry face at 13° . The explosive charge consisted of 15,400 pounds of 60, 75, and 80 percent low-freezing and high-velocity gelatin and was fired by a 220-volt circuit connecting all holes in parallel. Specially constructed cordeau, including insulated electric wires, was stretched from top to bottom of each hole, with a No. 8 cap at the top and another at the bottom of the hole. Thus, all holes were detonated at top and bottom simultaneously.

Three seismometer stations were established^{5/} behind the quarry face, stratigraphically higher than the collars of the holes. The three stations were in a straight line and were 617 feet (station A), 1,365 feet (station B), and 2,075 feet (station C) from the blast, horizontally. Station A was 140 feet above the quarry floor and set on clay overburden. Station B was 183 feet above the quarry floor and on an outcrop of a formation stratigraphically above the shot. Station C was 286 feet above the floor and set in a plowed field.

The first impulse reached station A in 0.04 second, B in .06 second, and C in .10 second.

From these data, the apparent speeds of propagation of the seismic wave were as follows: To A, 15,400, to B, 22,700, and to C, 20,750 feet per second.

Station B, on an outcrop, was used in calculating the speed of propagation of the seismic wave through the rock, which was 22,700 feet per second.

^{5/} A seismometer station consists of three seismometers, two of which record horizontal components and one the vertical.

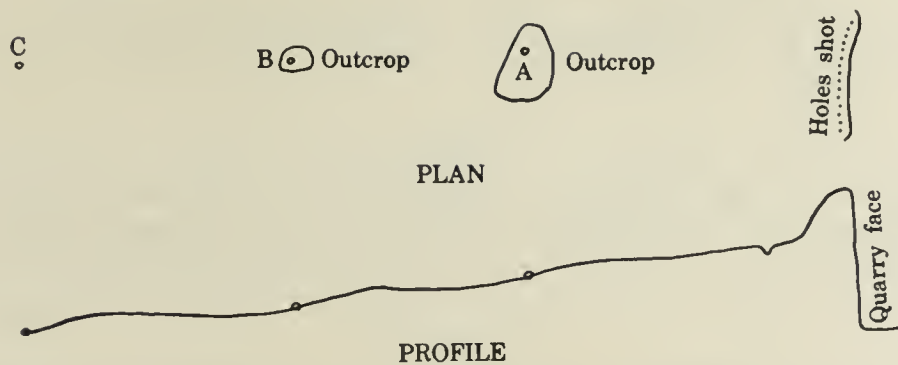
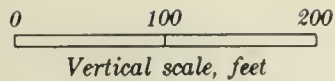
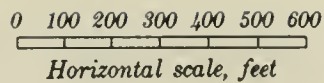


Figure 1.—Quarry B.

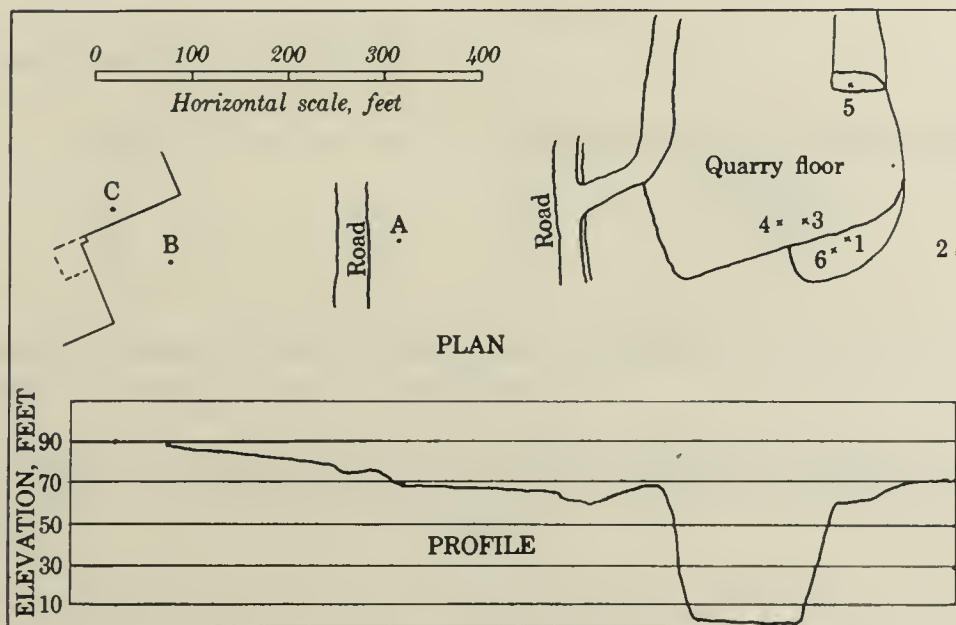


Figure 2.—Quarry C.

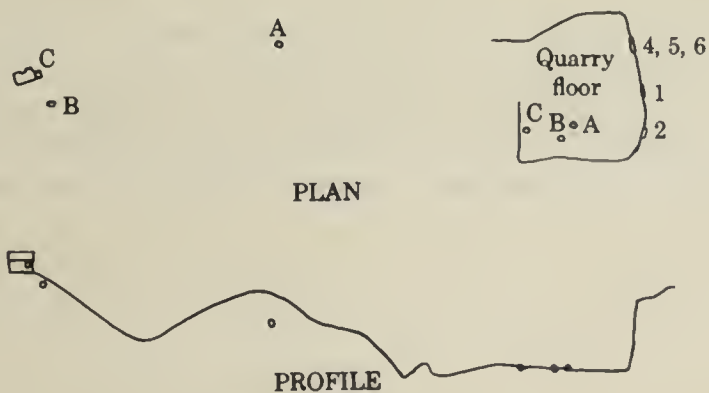
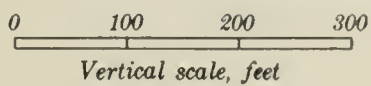
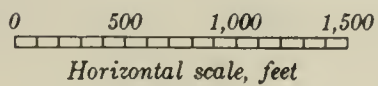


Figure 3.—Quarry E.

No measurements were made of the depth of overburden at stations A and C. If the speed of propagation through the overburden is assumed to be 3,000 feet per second, then at a rock speed of 22,700 feet per second the overburden at A is calculated to be 38 feet thick and at C 26 feet. At the quarry face the overburden was observed to be 20 feet thick.

A compressional air wave was recorded by disturbance of the oscillograph conveyance and of the seismometers at station A. This impulse was recorded at 0.085 second after the blast, equivalent to an average propagation speed of 7,250 feet per second at station A. It was not recorded at stations B or C.

The photographic record of the vertical seismometer at station A was imperfect and unreadable, hence the maximum resultant amplitude for that station can only be estimated from the records of the two horizontal instruments. The records for stations B and C were all readable.

From the photographic records, the greatest resultant motion at the seismometer stations was as follows:

<u>Station</u>	<u>Maximum resultant amplitude, inch</u>
A	0.058
B	.0100
C	.0210

The maximum resultant amplitude, as calculated, means the maximum movement from point of rest - single amplitude.

The frequency of vibrations at station B on the outcrop was 20 cycles per second for 0.5 second, while at A it was 12 for 0.8 second and at C 8 for 0.8 second.

Since stations A and C were on overburden, while B was on an outcrop, direct comparison of damping of the wave in distance cannot be made through all three stations, but it is notable that the damping from station A to station C is not directly proportional to the distances of these stations from the shot.

Quarry B

A single shot was made at this quarry (fig. 1) with the charge placed in 15 well-drill holes 6 inches in diameter and ranging in depth from 82 to 99 feet. The holes were drilled in biotite-gneiss of massive structure and without major faults or seams. The explosive charge was 11,500 pounds of 60-percent straight nitroglycerin dynamite and was fired by cordeau detonated by a single no. 6 cap on the trunk line. The cap was fired by a 45-volt battery.

Three seismometer stations were located in a line behind the quarry face at distances and elevations referred to the quarry floor, as follows:

<u>Station</u>	<u>Horizontal distance,</u> <u>feet</u>	<u>Vertical distance,</u> <u>feet</u>
A	640	+42
B	1,100	+25
C	1,650	+11

Stations A and B were on outcrops and station C was on clay overburden.

Station A received the first impulse 0.0525 second after the blast, station B 0.070 second, and station C 0.12 second. Since the first two stations were on outcrops they were used to calculate the speed of propagation of the seismic wave in the rock, which was found to be $12,200^{\pm}$ feet per second at A and $15,700^{\pm}$ at B.

The speed between stations A and B was calculated to be $26,300^{\pm}$ feet per second and is considered the rock speed. The slower apparent speeds between the shot and stations A and B are assumed to be caused by a discontinuity between the shot and station A.

No measurements were made of the depth of overburden at station C. If the speed through clay is assumed to be 3,000 feet per second, the depth of overburden at C is calculated to be 170 feet.

A compressional air wave was recorded by disturbance of the seismometers at station A 0.11 second after the blast, from which its speed was calculated to be 5,820 feet per second. This compression wave was not recorded at either station B or C. A second air wave was recorded at station A at 0.61 second calculating to a speed of 1,050 feet per second, or the speed of sound. This explains a sharp report heard following the main blast.

From photographic records the greatest resultant motion at the seismometer stations was as follows:

<u>Station</u>	<u>Maximum resultant amplitude, inch</u>
A	0.028
B	.016
C	.054

The frequency of vibrations at stations A and B was 20 cycles per second for 0.6 second. At C it was 7 cycles per second for 2.0 seconds.

Since stations A and B were both on outcrops while C was on overburden, a direct comparison of the damping of the wave through all three stations is not possible, but it is notable that the damping from station A to station B is directly proportional to the distances from the blast.

Freight-train record

With the instruments in the same position and without changing their sensitivity setting, an attempt was made to record the vibrations received from a passing freight train of 50 empty cars traveling about 40 miles per hour. For this test, the distances from the track to the stations were as follows:

<u>Station</u>	<u>Distance, feet</u>
A	1,140
B	1,600
C	2,150

No record was obtained, which shows that the vibration must have been less than 0.0001 inch at the stations.

Quarry C

At this quarry six shots were recorded. (fig. 2.)

Shot 1 was a charge in a single well-drill hole 6 inches in diameter and 61 feet deep. The hole was in limestone interfolded with shist, the beds dipping at 45° . It was loaded with 250 pounds of 50-percent nitro-starch explosive fired by electric detonator.

The quarry is of the pit type and the seismometer stations were on the surface in front of and across the quarry from the shot. The distances and elevations referred to the quarry floor were as follows:

<u>Station</u>	<u>Horizontal distance, feet</u>	<u>Elevation, feet</u>
A	451	+68
B	684	+88
C	740	+90

Stations A and B were on overburden and station C was on the concrete basement floor of a four-story brick building.

Station A received the first impulse 0.04 second after the blast, station B 0.05 second, and station C 0.06 second. Since all stations were on overburden, the speed of propagation through the underlying strata cannot be computed accurately, as no measurements were made of the thickness of the overburden.

From the photographic records the maximum resultant motion at the seismometer stations was as follows:

<u>Station</u>	<u>Amplitude, inch</u>
A	0.0030
B	.0054
C	.0029

The frequency of the vibrations ranged from 15 to 30 cycles per second at all stations; duration at all stations was 0.7 second.

In this case all stations were on overburden overlying the bed rock. C, however, being on the concrete basement floor, was not directly comparable to A and B. Furthermore, no measurements of overburden depth were made at A and B. A study of the amplitudes shows a damping from A to B closely proportional to the distance from the shot. From B to C, however, the wave was damped much more. This may be simply an increasing deceleration with distance, or it may be the result of loss of energy in overcoming the inertia of the concrete floor or transmission through it. One horizontal seismometer at station C gave an unsatisfactory record, which may account for some of this increased deceleration.

Shot 2

In this shot an attempt was made to record any possible vibrations transmitted to the strata by a blast in the overburden.

Ten holes 9 feet deep were shot to excavate a ditch in the surface. The explosive charge was 35 pounds and station A was 560 feet distant, with B and C approximately 800 feet.

No measurable amplitudes were recorded, due, possibly, to complete damping of the wave from the small shot before it reached the stations, to reflection from the rock surface and absorption, or to too insensitive setting of the seismometers. The movement, if any, was definitely less than .001 inch at the nearest station, A.

Shot 3

This shot was made in an attempt to record any vibrations that might be caused by a mudcap shot.

The explosive charge of 0.36 pound 30-percent gelatin was detonated on a large boulder 450 feet from station A, 683 from B, and 740 from C.

No record of a ground vibration was obtained. The air wave was received at station A in 0.41 second and at B in 0.64 second, from which the speed was calculated at 1,100 feet per second at station A and 1,070 at station B.

Shot 4

This shot was made in another attempt to obtain vibrations from a mudcap shot. In this case, 1.13 pounds of 30-percent explosive was shot 425 feet from station A, 660 feet from B, and 715 feet from C. No ground vibrations were recorded at any station, but each seismometer was disturbed by the air wave. The time to station A was 0.36 second, to B 0.585 second, and to C 0.61 second, from which the speed of propagation of the air wave was calculated at 1,180 feet per second at station A, 1,130 at B, and 1,170 feet per second at C.

Station C, in the basement, gave a maximum resultant amplitude of 0.00015 inch. This is interesting for comparison with another later shot and also because it indicates that the air wave striking the house can and does vibrate the foundation.

Shot 5

This shot consisted of 36 pounds of 50-percent nitrostarch explosive in six jackhammer holes. The holes were in a bench 45 feet above the quarry floor and were fired simultaneously. The distances from the stations were as follows:

<u>Station</u>	<u>Horizontal distance, feet</u>	<u>Vertical distance, feet</u>
A	480	+43
B	710	+63
C	755	+65

Station A received the first vibrations .045 second after the blast, station B .055 second, and station C .06 second. No propagation speeds are computed because all stations were on overburden.

The frequencies varied at all stations and ranged from 10 to 40 cycles per second for 0.6 second. The maximum resultant amplitudes were:

<u>Station</u>	<u>Amplitude, inch</u>
A	0.00108
B	.00082
C	.00035

It will be noted that the amplitude at station C was double that obtained at the same station from the air wave in shot 4, giving a rough comparison between the air wave and ground vibrations on the foundation of a structure.

Shot 6

This shot was a charge in a single well-drill hole 6 inches in diameter and 68 feet deep loaded with 300 pounds of 50-percent nitrostarch explosive and located 440 feet from Station A, 673 feet from B, and 730 feet from C.

Station A was 68 feet above the quarry floor, station B 88 feet, and station C 90 feet.

Station A received the first impulse in .04 second, station B in .05 second, and station C in .06 second. No propagation speeds were computed. The maximum resultant amplitudes were:

<u>Station</u>	<u>Amplitude, inch</u>
A	0.008 (estimated)
B	.0042
C	.0030

The vertical seismometer at station A gave a blurred record, hence could not be read accurately.

It will be noted that these amplitudes check, within the range of accurate reading, with those of shot 1 at this quarry.

The frequencies ranged from 10 to 40 cycles per second for 0.7 second.

General Discussion

From the results obtained at this quarry it is found that:

1. It is doubtful if blasts in the overburden set the underlying strata in appreciable vibration.
2. Mud-cap shots do not appreciably vibrate the subsurface strata but do transmit vibrations to the building foundation as a result of the effect of the air wave on the building.
3. Assuming a rock speed of 17,000 feet per second for this quarry, the depths of overburden were calculated as follows:

	<u>Depth, feet</u>		
<u>Station</u>	<u>Shot 1</u>	<u>Shot 2</u>	<u>Shot 3</u>
A	41	45	42
B	29	26	31
C	31	28	33

Quarry D

This quarry is in horizontally bedded limestone with a 30-foot face. Below the quarry floor lies another bed of limestone 40 to 100 feet thick, which, in turn, is underlaid with shale.

Thirty-four hammer-drill holes were drilled on the corners of equilateral triangles, 4 1/2 feet on a side, in the top bench of the quarry. The holes were 2 inches in diameter and 11 feet deep and were loaded with 232 pounds of 60-percent special gelatin dynamite. Charges were detonated electrically by no. 6 caps. All holes were shot simultaneously.

The seismometer stations were all on overburden behind the quarry face. Distances from holes and elevations above the quarry floor were as follows:

<u>Station</u>	<u>Horizontal distance, feet</u>	<u>Vertical elevation, feet</u>
A	443	+28
B	641	+43
C	882	+49

The first impulse was recorded at station A in 0.025 second, at B in 0.03 second, and at C in .045 second.

Assuming a rock speed of 24,000 feet per second, the depth of overburden was calculated to be 20 feet at station A, 10 feet at B, and 25 feet at C.

The frequencies at all stations were 30 cycles per second for 0.25 second. The maximum resultant amplitudes were as follows:

<u>Station</u>	<u>Amplitude, inch</u>
A	0.005 (estimated)
B	.004
C	.003 (estimated)

The vertical seismometers at stations A and C gave a blurred record, hence the amplitudes cannot be computed accurately. It is noted, however, that the amplitude diminished with increase of distance from the shot.

Quarry E

This quarry is a side-hill quarry with a 60-foot face of bedded limestone. (Fig. 3.) In quarrying operations, 5 feet of limestone is left for the quarry floor, below which lies a bed of shale 12 feet thick, which, in turn, overlies 40 feet of dolomite. Beyond the quarry area the limestone being quarried is overlain by another bed of limestone.

Six shots were made under variable conditions, described below.

Shot 1

This shot consisted of 32 jackhammer holes, 2 inches in diameter, charged with 34 pounds of 40-percent ammonia gelatin dynamite. They were connected in series and detonated simultaneously by electric current.

For this shot the seismometer stations were located at distances and elevations referred to the quarry floor as follows:

<u>Station</u>	<u>Horizontal distance, feet</u>	<u>Vertical distance, feet</u>
A	330	0.0
B	400	0.0
C	530	0.0

All stations were set on the limestone quarry floor. The shot was made on a bench resting on the quarry floor.

The first impulse was recorded at station A in 0.04 second, at B in 0.04 second, and at C in 0.045 second. Since all were on the ledge more or less tightly coupled with the shot, the speed of propagation was computed and found to be 10,000 \pm feet per second.

The frequency was the same at all stations, viz., 30 cycles per second for 0.25 second. The maximum resultant amplitudes recorded were:

<u>Station</u>	<u>Amplitude, inch</u>
A	0.002 (estimated)
B	.002
C	.001

It will be noted that although station B was 70 feet farther from the blast than station A, the amplitude recorded there was similar. This is due to faulty photographic record for station A.

A delayed disturbance was recorded on one of the horizontal seismometers at station C 0.52 second after the shot. This probably was an air wave, and its speed is computed as 1,020 feet per second.

Shot 2

The second shot consisted of 21 jackhammer holes charged with 115 pounds of explosive and drilled on the second bench in the quarry face.

The seismometer stations were in the same setting as for shot 1, and the distances and elevations referred to the bottoms of the holes were as follows:

<u>Station</u>	<u>Horizontal distance, feet</u>	<u>Vertical elevation, feet</u>
A	300	-15
B	350	-15
C	500	-15

The first impulse reached station A in 0.035 second, station B in 0.035 second, and station C in 0.045 second.

The frequency at all stations was 25 to 30 cycles per second for 0.25 second. The maximum resultant amplitudes were:

<u>Station</u>	<u>Amplitude, inch</u>
A	0.003 (estimated)
B	.007
C	.004

It will be noted that the amplitudes recorded for this shot are damped in almost direct proportion to the distance from the shot.

Unfortunately, one horizontal and the vertical seismometer at station A gave imperfect records, hence the amplitude at that station is estimated and may be inaccurate.

In connection with this shot, it is interesting to note that all seismometers recorded a slight disturbance at 1.92 second following the shot. Since this time interval cannot be ascribed to sound or air wave, it is interpreted to be a record of the shock caused by the rock from the blast striking the quarry floor. The shot was located from 15 to 30 feet above the quarry floor.

Shot 3

This shot was similar to shots 1 and 2, but a defective record was obtained because of mechanical failure of the photographic recording apparatus. It is mentioned here only because the delayed disturbance assigned to falling rock is again evident, although in this case the time interval cannot be read.

For shots 4, 5, and 6 the seismometer stations were moved to new positions at greater distances and to obtain comparative results between vibrations recorded in a brick structure and those taken on the ground surface.

Station A was placed on the surface of a small elevation in front of the quarry and roughly 1,500 feet distant from the face. Station B was on clay overburden. Station C was placed on the second floor of a two-story brick house close to station B.

Shot 4

Shot 4 consisted of a number of jackhammer holes drilled in the lower bench of the quarry and loaded with 42 pounds of 40-percent ammonia gelatin dynamite.

Two seismometer stations only were used for this shot. The horizontal distances and vertical elevations, as referred to the quarry floor, were as follows:

<u>Station</u>	<u>Horizontal distance, feet</u>	<u>Vertical elevation, feet</u>
A	1,470	+29
B	2,460	+57

Station A received the first impulse in 0.12 second and station B in 0.18 second.

Since the depth of overburden at station B was known to be approximately 10 feet, calculations were made to determine the speed of propagation, which was computed to be $14700 \pm$ feet per second for the underlying dolomite. Assuming a speed of 3,000 feet per second through the overburden, the depth of overburden at station A is 44 feet and at B 11 feet.

The frequencies at both stations were 20 cycles per second for 0.2 second at A and 0.5 second at B. The maximum resultant amplitudes were as follows:

<u>Station</u>	<u>Amplitude, inch</u>
A	0.00027
B	.00033

It will be noted that in this test the computed speed is considerably greater than that recorded for shots 1 and 2. Further, the amplitude at station B is greater than that at A. The higher speed of propagation would indicate a path through some medium other than the shale below the quarry floor. However to reach station A, the path must necessarily follow the shale or the underlying dolomite, as the small elevation on which this station was set is isolated from any other means of vibration transmission. Rock speed was calculated at 14,700 feet per second in the dolomite and 10,000 feet per second in the shale between the quarry floor and the dolomite. With these speeds the path of the wave was calculated to minimum time, and from the difference between such time and the recorded time the depth of overburden was calculated.

Shot 5

In this shot stations A and B remained the same as for shot 4, and station C on the second story of the building was in operation.

This shot consisted of 213 pounds of 40-percent gelatin in several jackhammer holes.

The distances and elevations of the stations from the shot were as follows:

<u>Station</u>	<u>Horizontal distance, feet</u>	<u>Elevation, feet</u>
A	1,470	+ 9
B	2,460	+37
C	2,500	+54

The first impulse reached station A in 0.12 second, station B in 0.185 second, and station C in 0.19 second.

The frequencies at all stations were uniform at 22 cycles per second for 0.3 second at station A and 0.5 second for B and C. The maximum resultant amplitudes were as follows:

<u>Station</u>	<u>Amplitude, inch</u>
A	0.0007
B	.0025
C	.0004

A record was obtained of a disturbance at station C at 2.43 seconds following the blast on the two horizontal seismometers followed by a similar but much smaller disturbance at 2.48 seconds on the vertical. Station B also recorded this disturbance, but to a still smaller degree. Station A was not affected. The resultant amplitude at station C of this second disturbance was 0.0007 inch, or nearly double that caused by the earth vibration.

From the time element, the speed of propagation of this second disturbance is calculated to be 1,040 feet per second, or approximately the speed of sound. Hence, it is assumed this second disturbance was caused by a percussion or sound wave striking the house. It is interesting to note that the vibration recorded at the house was picked up first by the horizontal instruments due to horizontal impulse, and the vertical element was not affected till 0.05 second later. This seems to confirm the conclusion that this was the effect of an air vibration.

The frequency of this air-wave disturbance was 6 cycles per second on the transverse, 25 cycles on the longitudinal, and 50 cycles on the vertical. These variations in frequencies evidently indicate the effect of the house construction.

Shot 6

This shot consisted of 163 pounds of 40-percent ammonia gelatin in a group of holes at the top of the quarry face. The seismometer stations were the same as for shots 4 and 5. The distances and elevations were as follows:

<u>Station</u>	<u>Horizontal distance, feet</u>	<u>Vertical elevation, feet</u>
A	1,470	-11
B	2,460	+17
C	2,500	+34

Station A received the first impulse at 0.12 second, station B at 0.180 second, and station C at 0.19 second.

The frequencies at all stations were 20 cycles per second for 0.3 second at A and 0.5 for B and C. The maximum resultant amplitudes were:

<u>Station</u>	<u>Amplitude, inch</u>
A	0.0007
B	.0028
C	.0004

A second disturbance was recorded at station C at 2.41 seconds after the blast corresponding to the air wave mentioned in discussion of shot 5. The amplitude, as recorded from this air vibration, was 0.0003 inch. The frequency was 7 cycles per second.

Discussion

1. The speeds of propagation in the rock for shots 1 and 2 were calculated to be 10,000 feet per second, while those for shots 4, 5, and 6 were 14,700 feet per second. The first group undoubtedly represents the speed through the quarry floor. The speed of the latter group represents a faster transmitting medium, probably the dolomite below the shale.

2. Since both large and small explosive charges were used in both groups of shots, the difference in speeds of propagation cannot be ascribed to amount of explosive charge.

3. The frequencies of the first group were higher than those of the second group. Seismometer stations in the latter group of shots were on overburden, while in the first group they were on the quarry floor. Frequencies in overburden are consistently less than on outcrop.

4. It is difficult to interpret amplitude values. Those at station B for shots 4, 5, and 6 were greater than at A, although station B was 1,000 feet farther from the shot. In shots 5 and 6 amplitudes at B were more than three times those at A. On the other hand, those at C were only one-fifth to one-eighth those at B but at about the same distance.

The amplitudes at C undoubtedly reflect some absorption of the ground movement by the building structure before the wave reached the seismometer station on the second floor.

A possible explanation of the greater amplitudes at B than A lies in the position of the charges in the quarry face. Shot 4, of 42 pounds, was in the lower bench and the charge was concentrated near the quarry floor. Shot 5, of 213 pounds, was in the middle bench, and shot 6, of 163 pounds, was at the top of the face. There was no geologic stratum directly connecting the shots with stations A and B in a straight line other than the shale and dolomite below the quarry floor. Around the perimeter of the quarry, direct connection would be made between the shots and station B by limestone ledge. Station A was isolated from this ledge except through the quarry floor.

Shot 4 was a minimum charge and, while of sufficient impact to vibrate the quarry floor and underlying shale, may not have exerted enough energy to set the underlying dolomite in active vibration.

Shots 5 and 6, however, evidently imparted sufficient force to vibrate the dolomite and carry the energy to station B.

6. It is interesting to note that the air vibrations caused greater movement than the ground vibrations at station C, that the horizontal seismometers received the impulse before the vertical, and that the initial movement of the house due to air vibration was away from the blast.

Quarry F

This quarry is in a formation consisting of alternate beds of limestone and shale. It is a pit quarry with a face 30 to 40 feet high.

Two seismometer stations were established in line in front of the quarry face but beyond the quarry pit. A third station was established beside the farther of these but below it on an abandoned quarry floor.

Two shots were made and recorded.

Shot 1

This shot consisted of 21 wagon drill holes 2 1/4 inches in diameter and 31 feet deep loaded with 619 pounds of 50-percent and 60-percent ammonia dynamite.

The distances and elevations of the stations were as follows: The distances are wave-path distances and the elevations are referred to the quarry floor.

<u>Station</u>	<u>Wave path distance, feet</u>	<u>Vertical elevation, feet</u>
A	666	+37
B	1,136	+38
C	1,136	+ 1

The first impulse was recorded at station A in 0.05 second, at station B in 0.065 second, and at station C in 0.06 second. Station C was on the abandoned quarry floor and it was assumed to be directly connected by ledge to the shot. From this, the speed of propagation in the ledge was calculated to be 18,900 feet per second. Assuming the speed through overburden to be 3,000 feet per second, the overburden depth at station A is calculated to be 44 feet and at B 15 feet.

The frequencies recorded at stations A and B, both on overburden, were 20 to 30 cycles per second, while those at C were 50 cycles per second. The duration was 0.3 second for all stations.

The maximum resultant amplitudes were:

<u>Station</u>	<u>Amplitude, inch</u>
A	0.005 (estimated)
B	.0075 (estimated)
C	.0005

The vertical seismometers at both stations A and B gave blurred records, hence the amplitude could not be calculated accurately at those stations.

Shot 2

This shot consisted of 33 wagon drill holes 50 feet deep and charged with 543 pounds of 50-percent and 60-percent ammonia dynamite.

The distances and elevations for this shot were as follows:

<u>Station</u>	<u>Distance, feet</u>	<u>Vertical elevation, feet</u>
A	520	+37
B	1,000	+38
C	1,020	+1

In this shot the holes bottomed at the quarry floor, whereas in the preceding shot they bottomed 6 feet above the floor.

The first impulse was felt at station A in 0.045 second, at station B in 0.065 second and at station C in 0.06 second. The speed of propagation as calculated to station C was 17,000 feet per second, and the overburden at A was 43 feet and at B 18 feet. This checks within the range of readable error with shot 1.

The frequencies recorded at stations A and B were 20 cycles per second for 0.5 second and 50 cycles at C for 0.2 second.

The maximum resultant amplitudes were:

<u>Station</u>	<u>Amplitudes, inch</u>
A	0.009
B	.008
C	.0014

This gives a typical example of the increased amplitude due to overburden. Stations B and C were, roughly, the same distance from the shot, but the amplitude at C (on rock) was only 17.5 percent of that at B, while the frequency was double.

Quarry G

This quarry is in a trap-rock flow and has two floors at different elevations. The upper quarry has a face 190 feet high. The face at the lower floor, 40 feet below, ranges from 30 to 150 feet in height. The trap flow is underlain with sandstone dipping 20° from the horizontal and 20° south of east.

The lower sandstone outcrops to the west in a bold escarpment rising from a glacial valley between trap-rock ridges.

Three seismometer stations were established on the glacial valley floor in a line north 87° west from the sandstone escarpment and were, therefore, on glacial fill, stratigraphically below the trap rock flow.

Three shots were fired at this quarry. The first was in the eastern end of the quarry on the lower floor and consisted of 12 "snake" holes, 27 feet long, drilled under a face 30 feet high. These holes were loaded with 700 pounds of 60-percent special gelatin. This shot was used as an indicator to check the sensitivity of the seismometer settings.

The second shot was a coyote tunnel shot in the 190-foot face of the upper quarry. The tunnel was charged with 19,500 pounds of 40-percent gelatin dynamite placed in nine pockets containing approximately 2,200 pounds each.

The third shot was a coyote-tunnel shot in the lower quarry face charged with 28,200 pounds of 40-percent gelatin dynamite.

The distances and elevations (referred to the position of the explosive charge) of the seismometer stations for the three shots were as follows:

Station	Shot 1		Shot 2		Shot 3	
	Horizontal distance, feet	Elevation, feet	Horizontal distance, feet	Elevation, feet	Horizontal distance, feet	Elevation, feet
A	1,800	-83	1,550	-123	2,280	-86
B	2,290	-86	1,840	-126	2,750	-83
C	2,780	-83	2,330	-123	3,240	-63

For all three shots the seismometer stations were located on overburden. They were identical for shots 1 and 2, but for shot 3 Station A was moved to a greater distance and became station C for this shot.

Stations A and B in the first locations were on flat surface. Station C was on the edge of a creek that bordered and ran through a swampy area. This station (C) became station B for the third shot, and station C for shot 3 was on a low hill beyond the swampy area.

Shot 1

Seismometers were set at such sensitivity as was expected would best record the impulses from the heavier charges in the coyote-tunnel blasts of shots 2 and 3, and no records were obtained of shot 1 because the sensitivity was too low.

Shot 2

The first impulse was recorded at station A in 0.16 second, at B in 0.20 second, and at station C in 0.22 second.

At all stations a high-frequency wave of 30 to 80 cycles was recorded as superimposed on a low-frequency wave of 4 to 10 cycles per second for a duration of about 1 second, following which the low-frequency wave continued for 3 seconds with alternate periods of high amplitude and quiescence.

The maximum resultant amplitudes were as follows:

<u>Station</u>	<u>Amplitude, inch</u>
A	0.017
B	.013
C	.012

Shot 3

The first impulse was recorded at station A in 0.24 second, at B in 0.26 second, and at C in 0.30 second.

Two instruments at station A gave erratic readings for unknown reasons, which makes the record somewhat questionable.

At station B the frequency ranged from 30 to 50 cycles superimposed on a 4-cycle wave for 0.80^{\pm} second. During the next 0.5^{\pm} second all instruments at this station were quiet. A low-frequency wave of 3.8 cycles started on the horizontal-longitudinal seismometer at 1.5^{\pm} seconds, on the horizontal-transverse at 2^{\pm} seconds, and on the vertical at 3^{\pm} seconds. This wave continued to 5 seconds on the first instrument but terminated at 3.75 seconds on the latter two.

At station C, a high-frequency wave of 30 cycles prevailed for 0.2^{\pm} second followed by a 6-cycle wave for 0.4^{\pm} second, after which no disturbance was recorded.

The maximum resultant amplitudes were as follows:

<u>Station</u>	<u>Amplitude, inch</u>
A	0.012 (estimated)
B	.012
C	.005

Disucssion

In interpreting results from shots 2 and 3 it must be borne in mind that station B, shot 2, coincided with station A, shot 3; that station C, shot 2, coincided with station B, shot 3; and that the path of the wave from shots 2 and 3 traveled through about the same thickness of trap rock before impinging on the sandstone and other lower strata.

The apparent velocities of propagation, in feet per second, as computed from blast time and time of arrival were as follows:

<u>Station</u>	<u>Shot 2</u>	<u>Shot 3</u>
A	8,400	9,500
B	9,200	10,600
C	10,600	10,800

All stations were on overburden of undetermined depth, hence speed of propagation through the underlying rock strata cannot be computed directly. From a time-distance curve it is estimated that the speed of propagation through the underlying strata was 11,000 \pm feet per second. The geologic character of the subsurface strata below the valley is not known, but from regional evidence it is presumed to be sandstone.

It is noteworthy that at station C, shot 2 (station B, shot 3), a high-frequency wave was received first, followed by a quiescent period, which, in turn, was followed by a low-frequency wave. This was duplicated to a lesser degree at station B, shot 2 (station A, shot 3), and at station A, shot 2. At station C, shot 3, however, the period of high frequency lasted only 0.2 second, followed by a low frequency of only 0.4 second, after which none of the three instruments was affected.

The authors tentatively interpret this phenomenon to indicate that stations A, B, and C of shot 2 and A and B of shot 3 were on a glacial fill consisting of unconsolidated material, such as gravel and sand, with considerable moisture present. The high-frequency wave propagated from the underlying stratum set this unconsolidated mass in vibration at its own critical frequency as a mass. This critical frequency, being low, accounts for the low-frequency wave recorded. Station C, shot 3, recorded similar high and low frequencies, but for very short duration, indicating a continuation of the glacial fill, but of much shallower depth, in which the low-frequency wave was absent.

If the amplitudes of all stations for both shots be corrected for distance, it will be found that those for shot 3 were 30 percent greater than for shot 2, whereas the charge for shot 3 was 50 percent greater. This indicates that the impetus due to charge alone does not bear a straight-line relationship, which checks with previous tests of a comparable nature.

Quarry H

This quarry was in a trap-rock flow and the seismometer stations were on the flank of the trap ridge and in the valley between the quarry ridge and a similar trap-rock ridge.

Three shots were fired and the vibrations from them recorded.

Shot 1

This shot consisted of 5,100 pounds of 60-percent and 75-percent gelatin dynamite charged in three well-drill holes 6 inches in diameter and 136 feet deep.

Three seismometer stations were established on a line roughly parallel to the quarry face and running southeasterly from the quarry.

The distances and elevations (referred to the bottoms of the holes) were as follows:

<u>Station</u>	<u>Horizontal distance, feet</u>	<u>Vertical elevation, feet</u>
A	1,900	+57
B	2,700	- 1
C	4,400	-19

The first impulse was received at station A in 0.125 second, at B in 0.165 second, and at C in 0.265 second.

The frequency for all stations was 28 cycles per second for 1.0 second.

The maximum resultant amplitudes were:

<u>Station</u>	<u>Amplitude, inch</u>
A	0.014 (estimated)
B	.004
C	.002

Shot 2

This shot consisted of 6,050 pounds of 60-percent and 75-percent gelatin dynamite charged in four well drill holes 6 inches in diameter by 147 feet deep.

For this shot station C remained as for shot 1 but was designated as station A for shot 2. Stations B and C were on the same line as before but at a greater distance from the shot. Stations B and C were on the valley floor between the trap-rock ridges. The distances and elevations were:

<u>Station</u>	<u>Horizontal distance, feet</u>	<u>Vertical elevation, feet</u>
A	4,510	-12
B	5,000	-13
C	5,960	-35

No blast time was recorded for this shot, due to mechanical difficulties, but the time interval between the receipt of the first impulse at station B and station C was 0.06 second and that between stations A and B was 0.05 second.

The initial frequency at all stations was 28 cycles per second. At station A this initial vibration lasted 0.25 second and was followed by a period of 0.5 second during which the vibration was erratic but throughout which the initial high frequency was discernible. At stations B and C the high-frequency wave was imposed upon a low-frequency wave of 5 cycles. The amplitude of the low frequency was unexpectedly large and lasted much longer than anticipated. In fact, the oscillograph record was stopped while the instruments were still vibrating. The duration was known to continue for 3 seconds. The maximum resultant amplitudes were:

<u>Station</u>	<u>Amplitudes, inch</u>
C	0.007 (estimated)
B	.0035
A	.0018

With the seismometer stations at the same points and set at the same sensitivity as for shot 2, an attempt was made to record the vibrations from a passing passenger train composed of an engine and two cars.

One horizontal instrument only gave a readable record. This was at station A, which was 130 feet from the track. The recorded amplitude was 0.00026 inch double amplitude or, 0.00013 single amplitude, as comparable with those from the shot.

Shot 3

This shot consisted of 7,500 pounds of 60-percent and 75-percent gelatin dynamite charged in 6 well-drill holes and fired simultaneously. The seismometer stations were the same as for shot 2. The distances and elevations were:

<u>Station</u>	<u>Horizontal distance, feet</u>	<u>Vertical elevation, feet</u>
A	4,010	-31
B	4,510	-32
C	5,470	-44

The first impulse arrived at station A in 0.245 second, at B in 0.295 second, and at C in 0.355 second. This checks the time interval between stations for shot 2.

Station A again recorded a high-frequency wave of 28 cycles per second for 0.6 second, being somewhat erratic over the last portion of the period.

Stations B and C, however, gave almost no indication of a high-frequency wave, but all instruments recorded a low-frequency vibration of 6 cycles that continued for approximately 5 seconds. The maximum resultant amplitudes were:

<u>Station</u>	<u>Amplitudes (inch)</u>
C	0.007 (estimated)
B	.004
A	.0036

Discussion

Since all stations were on overburden, the speed of propagation through the rock cannot be determined directly from the records. However, the time was plotted against distance and the speed calculated to be 17,000 feet per second in the trap rock.

For convenience, the explosive charge, horizontal distances from the shots, and wave amplitudes for the three shots are tabulated as follows:

Station	Shot 1			Shot 2			Shot 3		
	Explosive charge, pounds	Distance, feet	Amplitude, inch	Explosive charge, pounds	Distance, feet	Amplitude, inch	Explosive charge, pounds	Distance, feet	Amplitude, inch
A	5,100	1,900	0.014	6,050	4,510	0.0018	7,500	4,010	0.0036
B		2,700	.004		5,000	.0035		4,510	.004
C		4,400	.002		5,960	.007		5,470	.007

In comparing amplitudes in these tests, it is noted that station C, shot 1, and station A, shots 2 and 3, are identical in location. Shot 1, with 20 percent less explosive and at 2 percent less distance, gave nearly identical amplitude. A plot of an amplitude-distance curve under these conditions shows the amplitude to be damped rapidly at first but slowly at the distance involved. Therefore, the small increase in distance for shot 2 corresponds to a total distance in which most of the damping has already taken place. Hence, the increased charge has proportionately less tendency to increase the amplitude at this distance.

On the other hand, the 7,500-pound charge in shot 3, although only 25 percent greater than shot 2, doubled the amplitude. This, however, was for a distance 10 percent shorter. Here, then, occurs a combination of increased charge and decreased distance to cause the doubled amplitude under conditions comparable as to transmitting medium.

Continuing the comparison, it is interesting to note that in both shots 2 and 3 the amplitudes at stations B and C increase with distance rather than decrease, as was the case in shot 1. The change in transmitting medium is responsible for this. Stations B and C were on the valley floor, with station B on natural ground and C on man-made fill. It will be recalled that the frequencies at these two stations were high for a brief period at the beginning and then dropped to very low frequencies. This, again, illustrates the effect of a high initial vibration impinging upon a body of unconsolidated material and setting up therein vibrations at its natural period.

At the assumed rock speed of 17,000 feet per second, the depths of overburden were calculated as follows:

<u>Station</u>	<u>Shot 1</u>	<u>Shot 3</u>
A	6/ 6.0	27.0
B	18.0	90.0
C	13.0	99.0

Quarry I

This quarry is in intrusive diabase (trap) having a face averaging 70 feet in height. The shot was made in a coyote tunnel driven 50 feet under the face, with a right and left wing 47 and 60 feet long, respectively. The rock directly above the coyote wings was 90 feet above the quarry floor.

One wing was charged with three 2,000-pound pockets of 40-percent gelatin dynamite and the other with two 2,500- and two 3,000-pound pockets. In addition, there were five 24-foot snake holes to relieve one wing and three 22-foot holes to relieve the other. These were loaded with 250 pounds of 60-percent gelatin dynamite, making a total charge of 17,250 pounds. The charge was fired electrically, all caps being connected in parallel.

Three seismometer stations were located north 28° west of the quarry and behind the face. One station (A) was on overburden, another (B) was on crushed-rock fill forming the plant yard, and the third (C) was on the concrete floor of a garage and between stations A and B. The distances and elevations (referred to the quarry floor) were as follows:

6/ The overburden at this station was decayed vegetable soil, and the speed was assumed to be 500 feet per second; overburden speed at all other stations was assumed to be 3,000 feet per second.

<u>Station</u>	<u>Horizontal distance, feet</u>	<u>Vertical elevation, feet</u>
A	1,760	-110
B	1,830	-110
C	1,810	-110

Station B was 170 feet north of A.

The first impulse was received at station A in 0.16 second, station B in 0.18 second, and station C in 0.17 second. From these, the apparent velocities are calculated as 11,000 feet per second to station A, 10,450 to B, and 10,650 to C. Depth of neither overburden nor fill was known, hence speed of propagation through the underlying rock cannot be calculated directly.

At station A (on natural overburden) a frequency of 30 cycles per second was recorded on the three seismometers for a period of 0.2 second.

At all stations a low frequency of 3.3 cycles was recorded, which lasted for between 7 and 8 seconds.

This prolonged low frequency indicates that all stations were on loose or unconsolidated fill of considerable depth. The initial high frequency at station A represents the greater consolidation of the natural fill over that of the man-made fill at B and C. The maximum resultant amplitudes were as follows:

<u>Station</u>	<u>Amplitude, inch</u>
A	0.028
B	.068
C	.033

The horizontal-longitudinal instrument at station B gave a distorted record impossible to evaluate accurately. The other two instruments checked closely with those of the corresponding instruments at the other stations. It is concluded, therefore, that the maximum resultant at this station would check closely those at A and C, if this one instrument could be read accurately.

Discussion

Assuming a speed of 17,000 feet per second in the trap rock and 3,000 feet in the overburden, the depth of overburden at A was 170 feet, at B 208 feet, and at C 190 feet.

Quarry J

This quarry is in a trap-rock ridge with a face 70 feet high. The shot consisted of a coyote tunnel charged with 11,100 pounds of 40-percent gelatin dynamite fired electrically.

The seismometer stations were on overburden on a line parallel with the quarry face over the tunnel and on a flat valley floor. The distance and elevations (referred to the quarry floor) were as follows:

<u>Station</u>	<u>Horizontal distance, feet</u>	<u>Vertical elevation, feet</u>
A	2,500	-55
B	3,000	-55
C	3,600	-54

The first impulse reached station A in 0.17 second, station B in 0.20 second, and C in 0.26 second. The apparent propagation speeds were 14,700 feet per second to A, 15,000 to B, and 13,850 to C.

A high-frequency wave of 33 cycles per second was recorded at all stations lasting approximately 0.5 second. No low-frequency wave was recorded. The maximum resultant amplitudes were:

<u>Station</u>	<u>Amplitude, inch</u>
A	0.0046
B	.0036
C	.017

Discussion

The absence of a prolonged low-frequency wave indicates the absence of any deep unconsolidated glacial fill under the stations, and the apparently slow speed of propagation suggests that the valley is underlain with sandstone or shale. Depth of overburden was not determined, but if a speed of 15,500 feet per second is assumed for the underlying rock and 3,000 feet for the overburden, the depth of overburden at A is calculated to be 27 feet, at B 20 feet, and at C 84 feet.

Quarry K

This quarry is in limestone with a face 90 feet high. The upper half of the quarry consists of massive bedded stone, whereas the lower half contains thin bedded stone. The quarry floor consists of very thinly bedded mud rocks and shale.

The seismometer stations were located on overburden on a line running south from the quarry and in front of the quarry face. The first two stations were on a flat valley floor below the quarry floor and the third or farthest station was on the toe of the ridge across the valley.

Two shots were recorded.

Shot 1

This shot consisted of 13,400 pounds of 40-percent, 60-percent, and 75-percent gelatin dynamite charged in sixteen 6-inch by 95-foot well-drill holes and fired electrically. The distances and elevations (referred to the quarry floor) were as follows:

<u>Station</u>	<u>Horizontal distance, feet</u>	<u>Vertical elevation, feet</u>
A	2,000	-119
B	2,540	-123
C	3,630	-107

The first impulse reached station A in 0.215 second, B in 0.26 second, and C in 0.33 second.

At each station a high-frequency wave of 33 cycles per second was recorded for a period of 0.4 second, followed by a low-frequency wave of 5 cycles lasting 8 seconds. The maximum resultant amplitudes were:

<u>Station</u>	<u>Amplitude, inch</u>
A	0.025
B	.016
C	.0082

Shot 2

This shot consisted of 17,050 pounds of 40-percent, 60-percent, and 75-percent gelatin dynamite charged in 18 well-drill holes, 6 inches by 90 feet, adjacent to those of shot 1. The seismometer stations were in the same locations as for shot 1. The distances and elevations were as follows:

<u>Station</u>	<u>Horizontal distance, feet</u>	<u>Vertical elevation, feet</u>
A	2,090	-119
B	2,630	-123
C	3,720	-107

The first impulse reached station A in 0.22 second, B in 0.26 second, and C in 0.33 second.

At each station a high-frequency wave of 33 cycles per second was recorded for a period of 0.4 to 0.5 second, followed by a low-frequency wave of 5 cycles lasting 8 seconds or more. The maximum resultant amplitudes were:

<u>Station</u>	<u>Amplitudes, inch</u>
A	0.021
B	.014
C	.008

Discussion

Since all stations were on overburden of undetermined depth, the rate of propagation through the underlying shale could not be determined directly. From the data available, the apparent speeds of propagation, in feet per second, were:

	<u>Station A</u>	<u>Station B</u>	<u>Station C</u>
Shot 1	9,300	9,800	11,000
Shot 2	9,500	10,100	11,300

From this tabulation, it will be noted that the apparent speed of propagation was, roughly, 300 feet per second faster for shot 2 than for shot 1. A possible explanation of this increased speed lies in the fact that shot 2 was made on the day following shot 1, and on the day shot 2 was made it rained continuously all day and part of the previous night, so that the soil and subsoil were rather wet. It is believed that the saturated subsoil presented a denser medium for wave transmission and, therefore, induced a higher speed.

Both shots gave characteristic high initial frequency waves of short duration followed by low-frequency waves of long duration. This is characteristic of the effect of a high-frequency wave striking and setting a large mass of unconsolidated material to vibrating at its own frequency.

The inference to be drawn from this is that the valley contains a gravel fill of considerable depth derived from glacial or fluvial sources. In this case the path of transmission would not be a direct line through bedded shale in the valley bottom but probably reflected from the beds below the fill, hence the speed of propagation would be faster than that shown as the apparent speed because the path would be longer. Geologic reports for the region indicate the subsurface strata to be shale for considerable depth.

If a speed of 11,500 feet per second is assumed for the shale and 3,000 feet for the overburden and fill, the depths of overburden or fill for station A calculates to between 115 and 125 feet, while that for station B is 95 to 120, and for station C is 20 to 40.

For convenience, explosive charge, distances from the shots, and wave amplitudes are repeated in the following table:

Station	Shot 1			Shot 2		
	Explosive charge, pounds	Distance, feet	Amplitude, inch	Explosive charge, pounds	Distance, feet	Amplitude, inch
A	13,400	2,000	0.025	17,050	2,090	0.021
B	-	2,540	.016	-	2,630	.014
C	-	3,630	.0082	-	3,720	.008

It will be noted that in spite of the fact that the explosive charge for shot 2 was 27 percent more and the distances increased only 2.5 to 4.5 percent, the amplitudes of shot 2 were consistently lower than for shot 1, those at station A being 16 percent lower, at B 12.5 percent, and at C 2.5 percent lower.

No demonstrable reasons for this reduced amplitude from a heavier charge at equal distance can be given at this time. A possible explanation, however, may be advanced tentatively. The valley floor is assumed to be unconsolidated fill from the low-frequency wave recorded. Shot 2 was made while this fill was saturated with rain fall, while shot 1 was made during dry weather. Possibly the saturated condition of the fill enabled it to absorb and damp the wave more rapidly. The same condition may have been the cause of increasing the speed of transmission some 7 percent as already noted.

Quarry L

This quarry is located in gabbro-diorite with a face 60 to 70 feet high. Blasting is done by well-drill holes 6 inches in diameter by 60 to 70 feet deep.

Eleven tests were made to ascertain, if possible, the difference in directional effects of similar charges and for other purposes, which will be noted.

Shot 1

This shot consisted of two holes each loaded with 600 pounds of 60-percent gelatin dynamite.

The seismometer stations were on a line running south 45° west from the shot.

The distances and elevations (referred to the quarry floor) were as follows:

<u>Station</u>	<u>Horizontal distance, feet</u>	<u>Vertical distance, feet</u>
A	1,300	+47
B	1,480	+46
C	1,780	+67

At stations A and B the overburden was very thin, and at station A it was necessary that one seismometer be set on outcrop and a second had one leveling screw on rock and the other two on steel pins driven in the soil. At station C all seismometers were on steel pins, the length of which was approximately equal to the depth of the overburden.

The first impulse arrived at station A in 0.085 second, at B in 0.09 second, and at C in 0.11 second. Assuming all stations were on outcrops, the speed of propagation was calculated to be $16,000 \pm$ feet per second.

All stations recorded a high-frequency wave of 30 to 40 cycles per second for a duration of 0.4 second with no secondary low-frequency wave. The maximum resultant amplitudes were:

<u>Station</u>	<u>Amplitude, inch</u>
A	0.0046
B	.0026
C	.0027

Shot 2

This shot consisted of two holes, each loaded with 600 pounds of 60-percent gelatin and fired electrically as for shot 1.

The seismometer stations remained the same as for shot 1.

The distances and elevations remained as follows:

<u>Station</u>	<u>Horizontal distance, feet</u>	<u>Vertical distance, feet</u>
A	1,230	+47
B	1,400	+46
C	1,700	+67

The first impulse reached station A in 0.08 second, B in 0.085 second, and C in 0.105 second, from which the speed of propagation is calculated to have been $16,000 \pm$ feet per second.

All stations recorded a high-frequency wave of 30 to 40 cycles per second for a duration of 0.4 second. The maximum resultant amplitudes were:

<u>Station</u>	<u>Amplitude, inch</u>
A	0.0046
B	.0035 (questionable)
C	.0038

Discussion of shots 1 and 2

In both shots the speed of propagation and the frequency of vibrations checked closely.

At station B two instruments gave an erratic record, which obscured the accuracy of the reading.

These two tests may be said to check each other with reasonable accuracy.

Shot 3

This shot consisted of 1,200 pounds of 60-percent gelatin charged in two holes similar to shots 1 and 2.

The seismometer stations were placed in a line running south 62° east from the shots. The distances and elevations (referred to the quarry floor) were as follows:

<u>Station</u>	<u>Horizontal distance, feet</u>	<u>Vertical elevation, feet</u>
A	1,270	+ 4
B	1,300	- 8
C	1,300	+12

Station A was on an outcrop, station B was on a crushed-stone fill, and station C was on the attic floor (third floor) of a frame building, 3 feet from the north wall.

The first impulse reached station A in 0.07 second, station B in 0.085 second, and station C in 0.09 second. Since station A was on an outcrop, the speed of propagation was calculated from the time of arrival there at 18,000 feet per second. On this basis, the depth of fill at station B was 14 feet, assuming a speed of 1,000 feet per second through the rock fill. This is shown graphically in figure 4. With this assumption, the speed for the building structure calculates to 4,000 feet per second. The building foundation was set on the rock fill.

Station A recorded a high-frequency wave of 45 cycles per second for a period of 0.4 second with no secondary low-frequency wave.

Station B recorded a high-frequency wave of 45 cycles superimposed on a low-frequency wave of 16 cycles per second. The high-frequency wave terminated in 0.2 second, but the low frequency continued for another 0.9 second.

Station C recorded no high-frequency wave but did record a low-frequency wave of rather erratic nature varying from 12 to 20 cycles per second which lasted for a period of 2 seconds or more. The maximum resultant amplitudes were:

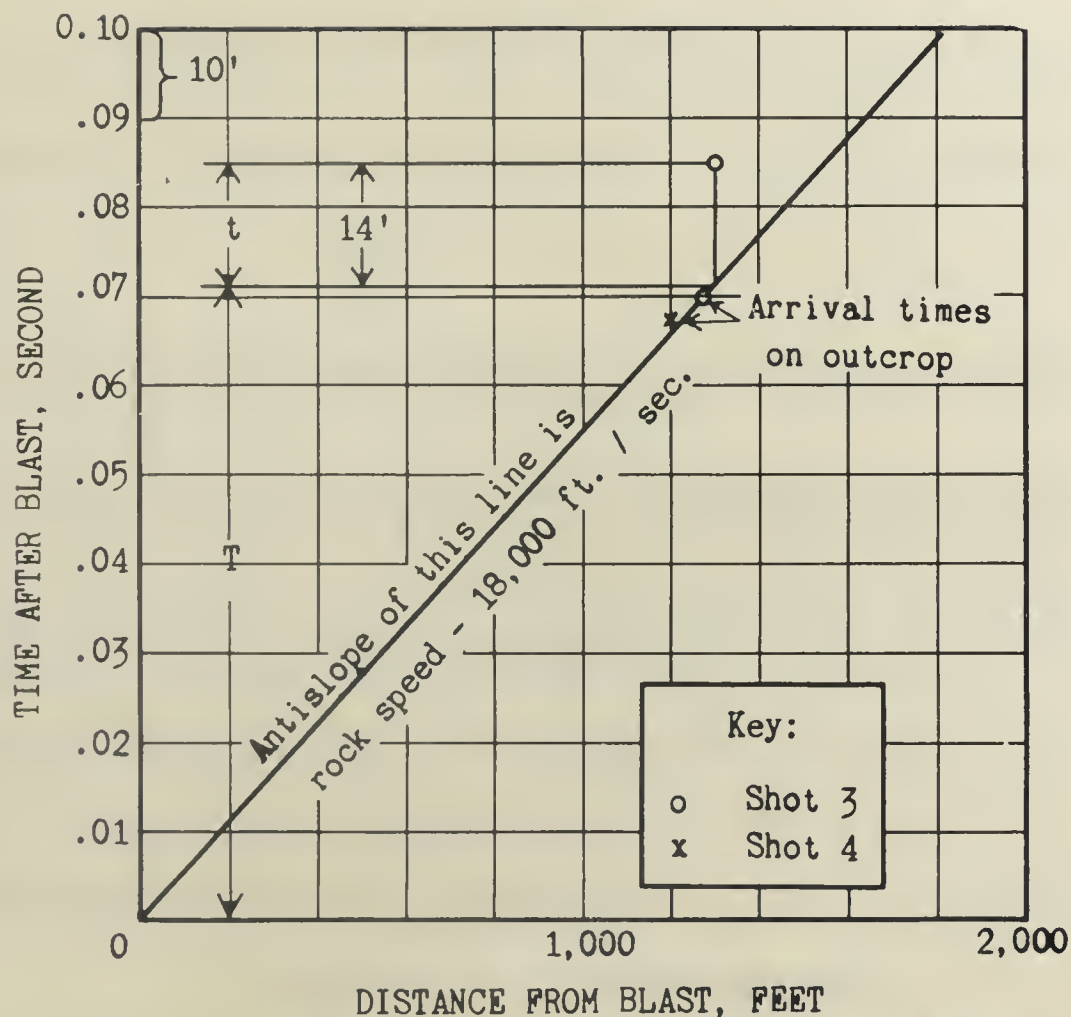


Figure 4.- Time-distance curve, Quarry L: t , Time required for impulse to traverse overburden. If speed through overburden is 1,000 feet per second, each unit of 0.01 second on the ordinate scale is equal to 10 feet. T , Time required for impulse to traverse rock.

<u>Station</u>	<u>Amplitude, inch</u>
A	0.0021
B	.0047
C	.0071

Shot 4

This shot duplicated conditions for shot 3 and the seismometer stations were at the same points. The distances and elevations were:

<u>Station</u>	<u>Horizontal distance, feet</u>	<u>Vertical elevation, feet</u>
A	1,200	+ 4
B	1,240	- 8
C	1,240	+12

The first impulse reached station A in 0.067 second, station B in 0.083 second, and station C in 0.086 second.

From the data obtained, the speed of propagation was calculated to be 18,000 feet per second and the depth of fill at station B 14 feet.

Station A recorded a 60-cycle high-frequency wave superimposed on a wave of 20 cycles for a period of 0.2 second on the vertical instrument. The horizontal instruments both recorded a wave of 25 cycles.

Station B recorded a wave of 16 cycles for 0.5 second.

Station C recorded an erratic wave of 10 to 16 cycles for 1.5 seconds.

The maximum resultant amplitudes were:

<u>Station</u>	<u>Amplitude, inch</u>
A	0.0028
B	.0077
C	.008

Discussion of shots 3 and 4

The speed of propagation checks at 18,000 feet for both shots.

The frequencies are fairly consistent as between stations, although there are some minor variations.

For convenience, the distances from the shot and wave amplitudes are reproduced in the table following:

Station	Shot 3		Shot 4	
	Distance, feet	Amplitude, inch	Distance, feet	Amplitude, inch
A	1,270	0.0021	1,200	0.0028
B	1,300	.0047	1,240	.0077
C	1,300	.0071	1,240	.008

It will be noted, first, that the amplitudes increase for each shot from station A to C or, in other words, for equal distances and explosive charge; the amplitude recorded on an outcrop is least, that on rock fill (overburden) is intermediate, and that on the upper story of a building set on fill is greatest. In each case the amplitude of vibration of the building was nearly three times that of the rock at the outcrop.

The amplitudes of shots 1 and 2 were greater than those for shots 3 and 4, for comparable distances, by 100 percent or more.

A similar unexplained anomaly is the lower speed recorded for shots 1 and 2 than for shots 3 and 4. A partial explanation for this, however, lies in the fact that between shots 1 and 2 and the stations lies a seam in the formation which may have affected the propagation of the wave.

Train vibrations

With seismometers set at the same stations as for shots 3 and 4, two attempts were made to record the vibrations transmitted by railroad trains passing the building in which station C was located.

For both tests station C was 120 feet horizontally from the center line of a double railway track. The seismometers of station C were 30 feet above the roadbed.

The sensitivity setting for the first and second tests remained the same as for the quarry shots.

In the first test a freight train passed at a rate of about 5 miles per hour.

In the second test a passenger train of 4 cars passed at 35 miles per hour.

No measurable amplitudes were recorded for either test, probably because the sensitivity setting was too low. The amplitude must have been less than 0.001 inch.

Building impact tests

Four tests were made to record vibrations imposed upon the building by mechanical impact.

A 4 x 4 x 40 inch piece of timber held in the hand was struck endwise against the midpoint of the north foundation wall of the building about 4 feet above the ground surface in the first two tests.

The sensitivity of the instruments at station C was increased and a record made.

The horizontal seismometer recording the component of motion in the direction the blow was struck (north and south) gave a double amplitude of 0.00058 inch, the transverse (east and west) 0.00086 inch, and the vertical 0.00106 inch, from which the resultant amplitude was calculated to be 0.00075 inch.

The frequency of the north-south seismometer was 50 cycles per second for 0.4 second, the east-west seismometer 60 cycles for 0.4 second, and the vertical 43 cycles for 1 second.

The test was repeated under the same conditions and the second record duplicated the first.

Two similar tests were then made, except that the blow was struck against the midpoint of the east wall. These two tests were again reproductions of each other.

The north-south seismometer registered a double amplitude of 0.00087 inch, the east-west of 0.00062 inch, and the vertical of 0.00069 inch, with a calculated resultant amplitude of 0.00060 inch.

In these two pairs of tests no exact measurement of the foot-pounds delivered by the blow was made, hence they cannot be compared exactly. It is noteworthy, however, that the larger of the two horizontal components was at right angles to the direction of the impact in both pairs of tests.

Walking test

A record was taken of the vibration caused by a 140-pound man walking past the seismometers. The record shows clearly the time of each step spaced at 0.56 second. The resultant amplitude was 0.00019 inch.

Discussion

Reference to the resultant amplitudes at station C of these tests shows that those from the timber impacts were roughly one-tenth the magnitude of those from the quarry blast, while that from a man walking was about one twenty-seventh.

Quarry M

This quarry is in gabbro-diorite with a face ranging in height from 110 to 135 feet.

Plans were made for four test shots, each composed of two well-drill holes each charged with 600 pounds of 60-percent gelatin dynamite. Each shot was then planned to contain 1,200 pounds of explosive. Unfortunately, the first shot broke the quarry face through an unseen slip carrying the upper portion of the next pair of holes with it and the explosive contained therein. The record of the second shot, therefore, is not comparable with those of the other three because the weight of explosive discharged is unknown. The first, third, and fourth shots, however, are directly comparable.

In these tests, the seismometer stations remained identical for all shots and the only difference in distances was due to the positions of the holes.

The seismometer stations were in a line north 37° east of and behind the quarry face. Distances and elevations (referred to the collars of the holes) were as follows:

Station	Shot 1		Shot 2		Shot 3		Shot 4	
	H. D.	V. El.	H. D.	V. El.	H. D.	V. El.	H. D.	V. El.
A	600	+184	635	+184	665	+184	700	+184
B	1,170	+ 97	1,210	+ 97	1,250	+ 97	1,280	+ 97
C	1,785	+ 34	1,825	+ 34	1,865	+ 34	1,900	+ 34

Path distances				
Station	Shot 1	Shot 2	Shot 3	Shot 4
A	628	662	690	724
B	1,174	1,213	1,253	1,284
C	1,785	1,825	1,865	1,900

The time of arrival of the first impulse at each station for each shot and the apparent speed of propagation were as follows:

Station	Shot 1		Shot 2		Shot 3		Shot 4	
	Time, sec.	Speed, ft./sec	Time, sec.	Speed, ft./sec	Time, sec.	Speed, ft./sec	Time, sec.	Speed, ft./sec
A	0.035	17,950	0.035	18,900	.035	19,700	0.04	18,100
B	.075	15,650	.075	16,200	.075	16,700	.08	16,050
C	.105	17,000	.105	17,400	.105	17,750	.11	17,250

Station A was on an outcrop, whereas B and C were on overburden of unknown depth.

In quarry L, in similar gabbro-diorite, the speed of propagation was determined to be 15,000 feet per second in one direction and 18,000 feet per second in another. In this quarry (M) the apparent speeds were all over 15,000 feet per second and it is evident that the speed through the rock itself is greater.

Station A was on an outcrop and the apparent speeds to that station are evidently rock speeds, the average of which is 18,660 feet per second. The difference in speeds at this station is due to the inability to read time at intervals of less than 0.005 second. The average, or 18,660 feet per second, is considered the rock speed for this quarry.

On this basis, the depths of overburden at stations B and C, as calculated from the rock speed, are as follows:

Calculated depths of overburden, feet.
Speed of propagation in overburden taken as 3,000 feet per second.

Station	Shot 1	Shot 2	Shot 3	Shot 4
B	36	30	23	33
C	28	21	15	24

The average depth at B is taken as 31 feet and at C as 22 feet.

The frequencies at stations A were consistently 50 cycles per second, at B all shots showed 16 to 50 cycles per second, and at C the frequency was consistently 30 cycles per second. The duration at all stations was 1/4 second or less, station A in all cases showing the shorter period.

Short duration at all stations indicates shallow overburden, and the comparatively low frequency wave at B may be interpreted to indicate a deeper overburden, thus supporting the preceding theoretical calculation of depths.

The explosive charge and maximum resultant amplitudes were as follows, in inches:

Shot	Explosive, pounds	Station A, inch	Station B, inch	Station C, inch ^{1/}
1	1,200	0.017	0.0076	0.002
2	?	.008	.0032	.0016
3	1,150	.013	.0057	.0037
4	1,300	.015	.0057	.0028

^{1/} Estimated.

Analysis of the preceding amplitudes, unfortunately, is subject to question because the vertical seismometer at station C gave an unreadable record, due to element vibration in the oscillograph.

It will be noted, however, that in all shots the amplitude decreased with increase in distance. At stations A and B the amplitudes for shots 1, 3, and 4 check within the range of readable error.

The amplitudes for shot 2 are lower in all cases, due to the loss of part of the explosive charge, as previously noted.

It will be noted, further, that the difference is greater between the amplitudes of A and B than between B and C. This is the expected result, due to normal damping with increasing distance.

If the amplitude-distance curves for shots 1 and 2, quarry L, are compared with the same curves for shots 1, 3, and 4, quarry M, they will be found to check within readable error. All five shots were in gabbro-diorite, and the explosive charge in each was about 1,200 pounds.

Quarry N

This quarry is in a volcanic complex of flintlike igneous rock, massive in character and with minor joint planes occurring in all directions. (Fig. 5.) The quarry face is semicircular, the face being advanced to the north. One major break or partly day-filled seam was observed in the east quarter of the face running north about 30° west.

The quarry is worked in benches by tripod drills that cut 2-inch holes averaging 20 feet deep. Fifteen to 30 holes were connected electrically and fired simultaneously for each shot.

Seismometer stations were placed in lines radially and otherwise around the quarry to test for various conditions as will appear. Nine shots were made and vibrations recorded.

Shot 1

Shot 1 consisted of 348 pounds of 60- and 75-percent gelatin dynamite shot in about the center of the quarry face.

The seismometer stations for this shot were strung out in a line running north from the quarry face. All stations were on outcrop east of the major seam. The horizontal distances were as follows:

<u>Station</u>	<u>Horizontal distance, feet</u>
A	330
B	585
C	875

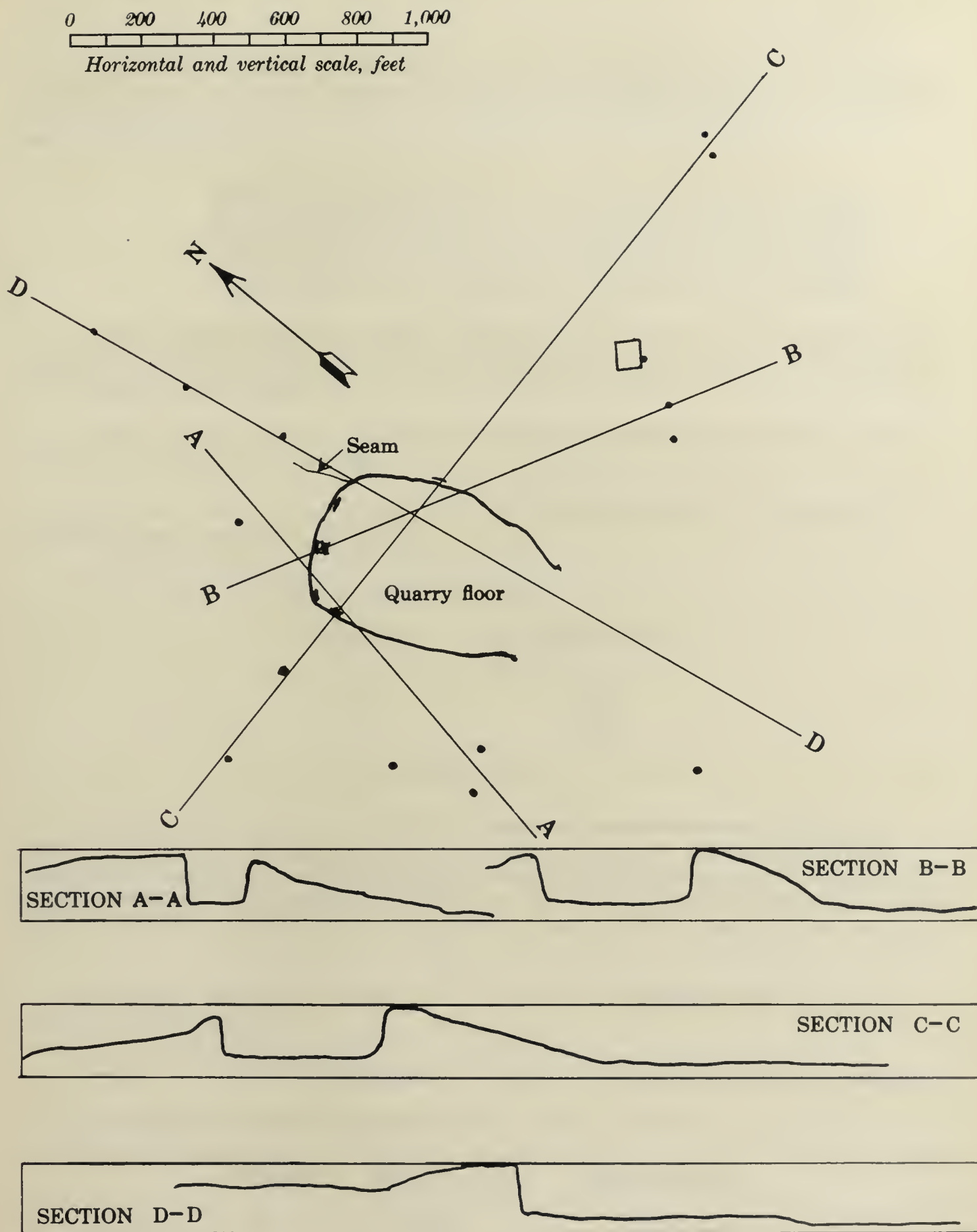


Figure 5.—Quarry N.

The first impulse reached station A in 0.0275 second, B in 0.0475 second, and C in 0.0675 second. Since these stations were all on outcrops, the speed of propagation could be computed directly from the record and were as follows:

From shot to station A12,000 feet per second.
 From shot to station B12,300 feet per second.
 From shot to station C13,000 feet per second.
 From station A to station B ...12,750 feet per second.
 From station B to station C ...14,500 feet per second.

This variable speed probably indicates the effect of the presence or absence of joint planes or other geologic conditions tending to decrease or increase the speed.

Frequencies at all stations were irregular and ranged from 20 to 50 cycles, but mostly 30 cycles.

The duration at stations A, B, and C was 0.2 second. No secondary low-frequency wave was recorded.

The maximum resultant amplitudes were:

<u>Station</u>	<u>Amplitude, inch</u>
A	0.01
B	.0027
C	.0022

Shots 2 & 3

Shots 2 and 3 were for tests designed to determine what effect the major seam had on the transmission of seismic waves. In both shots the seismometer stations remained in the same positions, but shot 2 was on the western side of the quarry and shot 3 on the eastern side east of the seam. In both cases one seismometer station was east of the seam and two west of it.

Station A of shot 1, east of the seam, was identical with station A of both shots 2 and 3.

Shot 2 consisted of 337 pounds of explosive and shot 3 of 335 pounds.

The distances from the shot were:

<u>Station</u>	<u>Shot 2, feet</u>	<u>Shot 3, feet</u>
A	510	435
B	440	490
C	370	565

The first impulse was received at the various stations for both shots as follows:

Station	Shot 2		Station	Shot 3	
	Time	Speed		Time	Speed
C	0.03	12,350±	A	0.035	12,400±
B	.035	12,600±	B	.040	12,250±
A	.045	11,350±	C	.045	12,550±

These results show the speeds between stations A and B, shot 2, to have been 7,000± feet per second and between the same stations, shot 3, to have been 11,000±, while the speeds between B and C for the same shots were 14,000± and 15,000± feet per second. Since the seam is known to lie between stations A and B in both shots, it seems reasonable to assume that the seam caused a reduction in the speed of propagation.

The frequencies at station A, shot 2, are somewhat erratic, but range between 30 and 50 cycles, duration 0.25 second.

At station B the frequency is 20 to 40 cycles for 0.25 second.

At station C the frequency is 30 cycles for 0.25 second.

At station A, shot 3, the frequencies are again erratic, ranging from 30 to 50 cycles for 0.25 second.

At station B the frequency varies from 25 to 50 cycles for 0.25 second.

At station C two instruments gave a foggy record but the third had a frequency of 20 to 50 cycles for about 0.1 second.

The maximum resultant amplitudes were:

<u>Station</u>	<u>Shot 2, inch</u>	<u>Station</u>	<u>Shot 3, inch</u>
C	<u>1</u> /0.006	A	0.008
B	.005	B	.0057
A	.0033	C	<u>1</u> /0.006

1/ Estimated.

It is unfortunate that mechanical difficulties at station C render the record there inaccurate.

In these three shots the average speed of propagation through the rock was 12,300 feet per second and all stations were north of the quarry.

Shots 4 and 5

Shots 4 and 5 were made with identical locations of the seismometers but with different explosive charges. The object of this test was to

determine, if possible, the effect of different explosive charges, keeping other conditions as nearly identical as possible.

Shot 4 consisted of 330 pounds of explosive and shot 5 of 183 pounds. Both shots were made on the west side of the quarry.

The seismometer stations were all on outcrops and were on the vertices of a triangle. The distances from shots were as follows:

<u>Station</u>	<u>Shot 4</u>	<u>Shot 5</u>
A	555	615
B	455	510
C	635	690

The time of arrival of the first impulse and the calculated speeds of propagation were as follows:

<u>Station</u>	<u>Shot 4</u>		<u>Shot 5</u>	
	<u>Time</u>	<u>Speed</u>	<u>Time</u>	<u>Speed</u>
A	0.035	15,850±	0.0375	16,400±
B	.03	15,150±	.0325	15,700±
C	.04	15,900±	.04	17,250±

Since these stations were not in line, computations cannot be made for speed between stations.

The frequencies range from 30 to 50 cycles per second for all stations for both shots and the duration was 0.2 second.

The maximum resultant amplitudes were:

<u>Station</u>	<u>Shot 4</u>			<u>Shot 5</u>		
	<u>Explosive, pounds</u>	<u>Distance, feet</u>	<u>Amplitude, inch</u>	<u>Explosive, pounds</u>	<u>Distance, feet</u>	<u>Amplitude, inch</u>
A	330	555	0.0020	183	615	0.0018
B	-	455	<u>1/</u>	-	510	.0026
C	-	635	.0021	-	690	.0012

1/ Records unreadable from two instruments.

This test must be considered negative because the amplitudes recorded are nearly as great for the smaller explosive charge as for the larger. Shot 5 shows consistent damping with increase in distance, but shot 4 shows greatest amplitude at the greatest distance, and the amplitudes at station A (intermediate in distance) for both shots are nearly identical.

Shot 6

This shot consisted of 355 pounds of explosive shot in the center of the quarry face.

The seismometer stations were moved to points southeast of the quarry. This setup was made to ascertain the difference between the vibrations transmitted to an outcrop, to overburden, and to the concrete foundation of a building, all at approximately equal distances from the shot and within a short distance of each other. The distances from the shot were:

<u>Station</u>	<u>Distance, feet</u>
A	1,125
B	1,145
C	1,140

Station A was on an outcrop, station B on overburden, and station C on a concrete apron structurally tied to the foundation of a brick building.

The first impulse arrived at station A in 0.075 second, at B in 0.08 second, and at C in 0.08 second. Since station A was on an outcrop, this station was used to determine the speed of propagation through the rock. The speed was calculated to be $15,000 \pm$ feet per second. Applying this speed to stations B and C, the overburden at both is calculated to be 11 feet deep.

The frequency obtained at station A was 30 to 50 cycles for a period of 0.25 second.

At B the frequency was 20 cycles for a period of 0.8 second.

At C the frequency was 10 to 15 cycles for a period of 1.5 second.

The high frequency and short duration at station A are characteristic of vibrations received on outcrops. The reduced frequency and lengthened duration at B are again characteristic of vibrations received on overburden. At C, again, the frequency was reduced and the period lengthened, indicating a possibility that the ground vibrations were transmitted to the building and set it in vibration, thus prolonging the duration.

The maximum resultant amplitudes were:

<u>Station</u>	<u>Amplitude, inch</u>
A	0.00085
B	.00245
C	.00375

This test again checks previous tests, in that amplitudes are least on outcrops, greater on overburden, and usually increased on structures founded on overburden.

Shot 7 and 8

Shot 7 was made in another quarry in the same kind of stone but three-quarters of a mile away. The shot consisted of 1,000 pounds of 75-percent gelatin dynamite fired electrically.

Shot 8 consisted of 203 pounds of explosive fired in the center of the original quarry face.

The seismometer stations were the same for both shots. Station A was on the same concrete apron as station C of shot 6. Station B was on an outcrop and station C was on overburden. The distances from shots, in feet, were:

<u>Station</u>	<u>Shot 7</u>	<u>Shot 8</u>
A	4,550	1,090
B	4,730	1,560
C	4,700	1,590

No blast time was recorded for shot 7 because it was impractical to stretch a wire to the second quarry. Therefore, no speed of propagation can be calculated for this shot.

The first impulse from shot 8 reached station A in 0.08 second, B in 0.10 second, and C in 0.103 second.

Since station B was on an outcrop, it is used to calculate the speed of propagation, which was 15,600 feet per second. Using this speed, the depth of overburden at A was calculated to be 30 feet and at C 3 feet.

It will be noted that rock speeds for shots 6 and 8 were 15,000 and 15,600 feet per second and all stations were east of the quarry.

The depth of overburden found at station A does not check with that found for C in shot 6, but it is within the range of readable time interval shown on the record.

The frequency at station A was erratic, ranging from 15 to 50 cycles for 1 second.

The frequency at station B was 40 to 50 cycles for 0.5 second.

The frequency at station C was 40 to 50 cycles for 0.5 second.

The high frequency and short duration at B confirm results found previously on outcrops. The similar result at C confirms the thin depth of overburden calculated. The low frequency and long duration at A check the results at C, shot 6.

The maximum resultant amplitudes were:

Station	Shot 7			Shot 8		
	Explosive, pounds	Distance, feet	Amplitude, inch	Explosive, pounds	Distance, feet	Amplitude, inch
A	1,000	4,550	0.00016	203	1,040	0.00225
B	-	4,730	.00008		1,560	.0007
C	-	4,700	.00011		1,590	.0012

These amplitudes strikingly confirm the results obtained from shot 6 in comparing vibrations received by outcrop, overburden, and structure.

Shot 9

This shot consisted of 442 pounds of explosive fired from the east center of the quarry face.

The seismometer stations were moved to the west side of the quarry to check the comparison between vibrations received on outcrop and overburden on this side of the quarry.

Stations A and C were in line running west from the shot. Station B was on the same outcrop as station B, shots 4 and 5. The distances from shots were as follows:

<u>Station</u>	<u>Distance, feet</u>
A	470
B	730
C	740

The first impulse arrived at station A in 0.04 second, at B in 0.045 second, and at C in 0.054 second.

Since station B was on outcrop it is used to determine the speed of propagation, which was $16,200 \pm$ feet per second. Using this speed, the depth of overburden at station A calculates to 33 feet and at C to 25 feet. It will be noted that rock speeds for shots 4, 5, and 9 averaged $16,000 \pm$ feet per second and all stations were west of the quarry.

The frequency at station A was 25 to 30 cycles for a period of 0.5 second. At B it was 40 cycles for 0.2 second and at C 20 to 40 cycles for a period of 0.5 second. The maximum resultant amplitudes were:

<u>Station</u>	<u>Amplitude, inch</u>
A	<u>1/</u> .02
B	.0025
C	.0095

1/ Estimated.

When located, it was assumed station A was on thinner overburden than station C. Speed calculations indicated the reverse to be true.

The oscillograph record shows the instruments at station A were vibrated at much greater amplitude than those at C, but the record cannot be read accurately because the curve extended beyond the edge of the photographic paper. The amplitudes recorded, however, show that those obtained on the outcrop were less than those obtained on the overburden.

It is interesting to note that the amplitude obtained at station B, shot 9, was almost identical with that obtained at the same point for shot 5, although shot 9 was charged with 442 pounds of explosive whereas shot 5 had only 183 pounds. The distance for shot 9, however, was nearly 50 per cent greater.

Station A was close to a street and the erratic behavior of the instruments there might have been due to the overburden.

Shot 10

Only one seismometer station was used for this shot. The purpose of the test was to ascertain the effect of vibrations on a large area of salt-water marsh near the quarry.

The station was south of the quarry and the shot, consisting of 500 pounds of explosive, was fired at the center of the quarry face.

The station was 1,330 feet from the shot. The first impulse was received on the vertical seismometer in 0.105 second. The two horizontal instruments received their first definite impulse in 0.29 second. The frequency on the vertical instrument was 14 cycles and on the horizontal 20 cycles.

The amplitude on the vertical instrument was only one-third that on the two horizontals, which were nearly equal. The maximum resultant amplitude was 0.0031.

The apparent speed of propagation was 12,650 feet per second. If the speed through the rock under the swamp is assumed to have been 15,000 feet per second, as recorded in previous tests, the depth of overburden at the station calculates to 50± feet.

Attention is drawn to the record showing that rock speeds to the north were 12,000± feet per second, to the east 15,000±, and to the west 16,000±, indicating the probability of geologic conditions affecting the speed of propagation.

Quarry O

This quarry had a face 100 feet high in trap rock overlying sandstone, which dipped to the east at 45°. The strike of the ledge was north 30° west.

The seismometer stations were all on overburden. Station A was just below the escarpment of the underlying sandstone, B was roughly 400 feet at right angles to the strike of the sandstone and below the escarpment, and C was 850 feet from the escarpment. All stations were on a flat valley floor and northwest of the shot.

The shot consisted of two coyote-tunnel blasts, one containing 16,800 pounds and the other 25,000 pounds of 40-percent gelatin dynamite fired simultaneously with a 550-volt alternating current.

The distances and elevations (referred to the quarry floor) were as follows:

<u>Station</u>	<u>Horizontal distance, feet</u>	<u>Vertical elevation, feet</u>
A	3,075	- 80
B	2,930	-101
C	2,925	-121

The first impulse reached station A in 0.20 second, B in 0.22 second, and C in 0.23 second. The apparent speeds of propagation to the three stations were: A 15,400, B 13,300, and C 12,700 feet per second. Since the wave must travel partly through trap and partly through sandstone in addition to passing through overburden of unknown depth at each station, and because of the steep dip of the sandstone, the actual path of the wave to each station is problematical and there are not sufficient data at hand to accurately calculate rock speeds.

The frequencies at all stations were somewhat erratic but predominately 5 cycles per second for 1.2 seconds. This indicates rather deep loose overburden set in vibration at its own frequency. No high-frequency waves were recorded.

The maximum resultant amplitudes were:

<u>Station</u>	<u>Amplitude, inch</u>
A	0.033
B	.020
C	.043

The record of one instrument at station C was difficult to read, hence the amplitude at that station is estimated.

It is significant, however, that station A received an impetus of higher amplitude than station B, although at greater distance. The wave also reached A before B. This indicates that the wave probably traveled a shorter distance through the underlying sandstone in its path to A than to B or C.

Assuming speeds of 17,000 for trap and 11,000 for sandstone, as previously determined in quarries H and G, and solving for the path of shortest time through the rock, the lengths of the wave paths to the three stations are calculated as follows:

<u>Station</u>	<u>Path, feet</u>	
	<u>In trap</u>	<u>In sandstone</u>
A	3,000	100
B	2,500	600
C	1,900	1,150

From these the overburden is calculated to be 42 feet at A, 66 feet at B, and 33 feet at C.

Quarry P

This quarry was in gabbro-diorite with a circular face. The shooting was done with snake holes drilled from the floor under the 10- to 15-foot quarry face. Eight shots were made and recorded and four other tests made, as discussed later.

The tests on the eight shots are divided into two groups. In one the seismometer stations were in a line running east from the quarry. In this group three seismometer stations were used for each of the two shots. For three other shots, one station only was used.

In the second group, two stations only were used for each of three shots. The stations were in a line running south from the quarry.

The first group was planned to determine the effect of distance on reduction of amplitude;

In the second group an attempt was made to measure the speed of the air wave initiated by the blast.

Shots 1 and 2

These shots consisted, respectively, of 80 pounds of 40- and 60-percent gelatin dynamite and 100 pounds of 40- and 60-percent gelatin dynamite.

The seismometer stations were in line east of the quarry at distances and elevations (referred to quarry floor) as follows:

Station	Shot 1		Shot 2	
	Horizontal distance, ft.	Elevation, ft.	Horizontal distance, ft.	Elevation, ft.
A	870	+ 2	932	+ 2
B	1,500	+ 23	1,525	+ 23
C	1,780	+118	1,820	+118

Station A was on resilient marshy soil, station B on firm overburden, and station C on sandy overburden.

The time of arrival of the first impulses and the apparent speeds are:

Station	Shot 1		Shot 2	
	Time	Speed	Time	Speed
A	0.055	15,800	0.055	16,950
B	.090	16,650	.090	16,950
C	.115	15,500	.110	16,550

Since all stations were on overburden, the speed of propagation through the underlying rock cannot be calculated directly.

Station A recorded frequencies of 20 cycles per second on the two horizontal instruments for a period of 0.4 to 0.8 second, but the vertical seismometer for both shots gave frequencies of 50 to 80 cycles for only 0.1 second.

Station B showed frequencies of 30 cycles for both shots on the horizontal instruments for 1.0 second or more. The vertical instrument gave a brief initial frequency of 80 cycles apparently superimposed on a 30-cycle wave for 0.8 second.

At station C, the horizontal instruments for both shots showed frequencies from 30 to 40 cycles while the vertical gave a high-frequency wave of 60 to 80 cycles on a 30-cycle wave for 0.8 second. The maximum resultant amplitudes were:

Station	Shot 1			Shot 2		
	Explosive charge, pounds	Distance, feet	Amplitude, inch	Explosive charge, pounds	Distance, feet	Amplitude, inch
A	80	870	0.0013	100	932	0.0012
B	-	1,500	.0008	-	1,525	.00055
C	-	1,780	.0006	-	1,820	.00035

Shot 3

For this shot a single station was set up on overburden 2,130 feet from the quarry and 125 feet above the quarry floor. The explosive charge was 100 pounds of 60-percent gelatin.

Shot 4 was 100 pounds of 60-percent gelatin and was recorded at a single station 4,160 feet from the quarry and 167 feet above the floor.

Shot 5 was 100 pounds of 60-percent gelatin and was recorded by a single station 7,440 feet from the quarry and 19 feet below the quarry floor.

No time of blast was recorded for these three shots, hence no speed of propagation can be computed. The frequencies recorded were as follows:

<u>Instrument</u>	<u>Shot 3</u>	<u>Shot 4</u>	<u>Shot 5</u>
Horizontal	20-30	40-50	40-50
Vertical	50-60	40-100	50

The maximum resultant amplitudes were:

<u>Shot 3</u>		<u>Shot 4</u>		<u>Shot 5</u>	
<u>Distance</u>	<u>Amplitude, in.</u>	<u>Distance</u>	<u>Amplitude, in.</u>	<u>Distance</u>	<u>Amplitude, in.</u>
2,130	0.0008(?)	4,160	0.00026	7,440	0.000055

Induced Vibration Test

With the instruments set up for shot 5, a comparative test was made by a man weighing 140 pounds jumping off a box and landing on his heels. The box was 18 inches high and was placed 40 feet from the seismometers.

Two jumps were recorded, the maximum resultant amplitudes of which were 0.00024 and 0.00020 inch, respectively, or approximately the same as for the 100-pound shot at 4,160 feet distance. The frequency was 50 cycles.

Air-Blast Tests

In the second group of tests a vane was fastened to the wooden cover of one seismometer so that any air movement caused by the blast would move the vane and box horizontally. No attempt was made to measure the amplitude of the air wave since there was no means of calibrating the instrument for air waves. The time of arrival only was noted, from which the speed of the air wave was computed.

Before recording a shot with the air vane, two records were made, one by tapping the vane and one by plucking. Both gave irregular curves with frequencies of the order of 15 cycles.

Shots 6, 7, and 8

The distances, explosive charges, and time of arrival of the first impulse were as follows:

Station	Shot 6			Shot 7			Shot 8		
	Explosive charge, pounds	Distance, feet	Time, second	Explosive charge, pounds	Distance, feet	Time, second	Explosive charge, pounds	Distance, feet	Time, second
A	80	135	0.03	100	180	0.055	100	210	0.07
B	-	495	.05	-	566	.060	-	422	.045

Station A in these three shots recorded only the time of arrival of the air wave. The station was located on a loose rock pile to insure insulation from ground vibrations. For shot 8 it was moved to the edge of the quarry on outcrop.

Station B was the same for all three shots on overburden.

Station A, shots 6 and 7, was 39 feet above the quarry floor. For shot 8 it was 15 feet above.

Station B was 25 feet above the floor on overburden.

The speed of the air wave and the apparent speed of the ground wave were, in feet per second:

Station	Shot 6		Shot 7		Shot 8	
	Distance, ft.	Speed	Distance, ft.	Speed	Distance, ft.	Speed
A, air wave	135	4,500	180	3,270	210	3,000
B, ground wave	-	9,900	-	9,400	-	9,400

The frequencies at station B for shots 6 and 7 were 40 cycles and for shot 8, 30 cycles. The maximum resultant amplitudes were:

	Shot 6			Shot 7			Shot 8		
	Explosive, lbs.	Distance, ft.	Amplitude, in.	Explosive, lbs.	Distance, ft.	Amplitude, in.	Explosive, lbs.	Distance, ft.	Amplitude, in.
Station B	80	495	0.0012	100	566	0.0017	100	422	0.0025

These three shots show the air wave diminishing rapidly in speed with distance, and it apparently damped out before reaching station B since no record of it was obtained there.

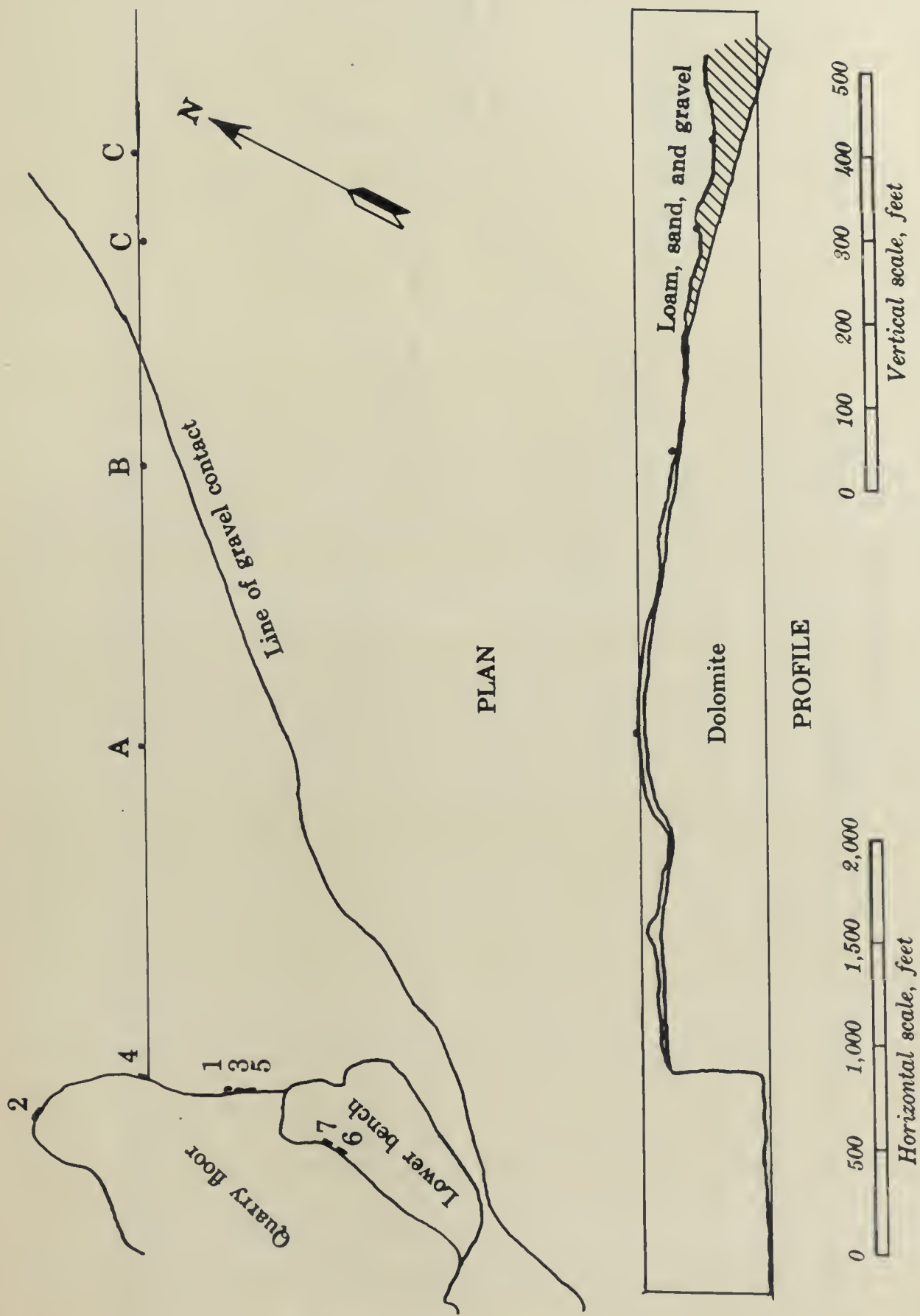


Figure 6.- Quarry Q.

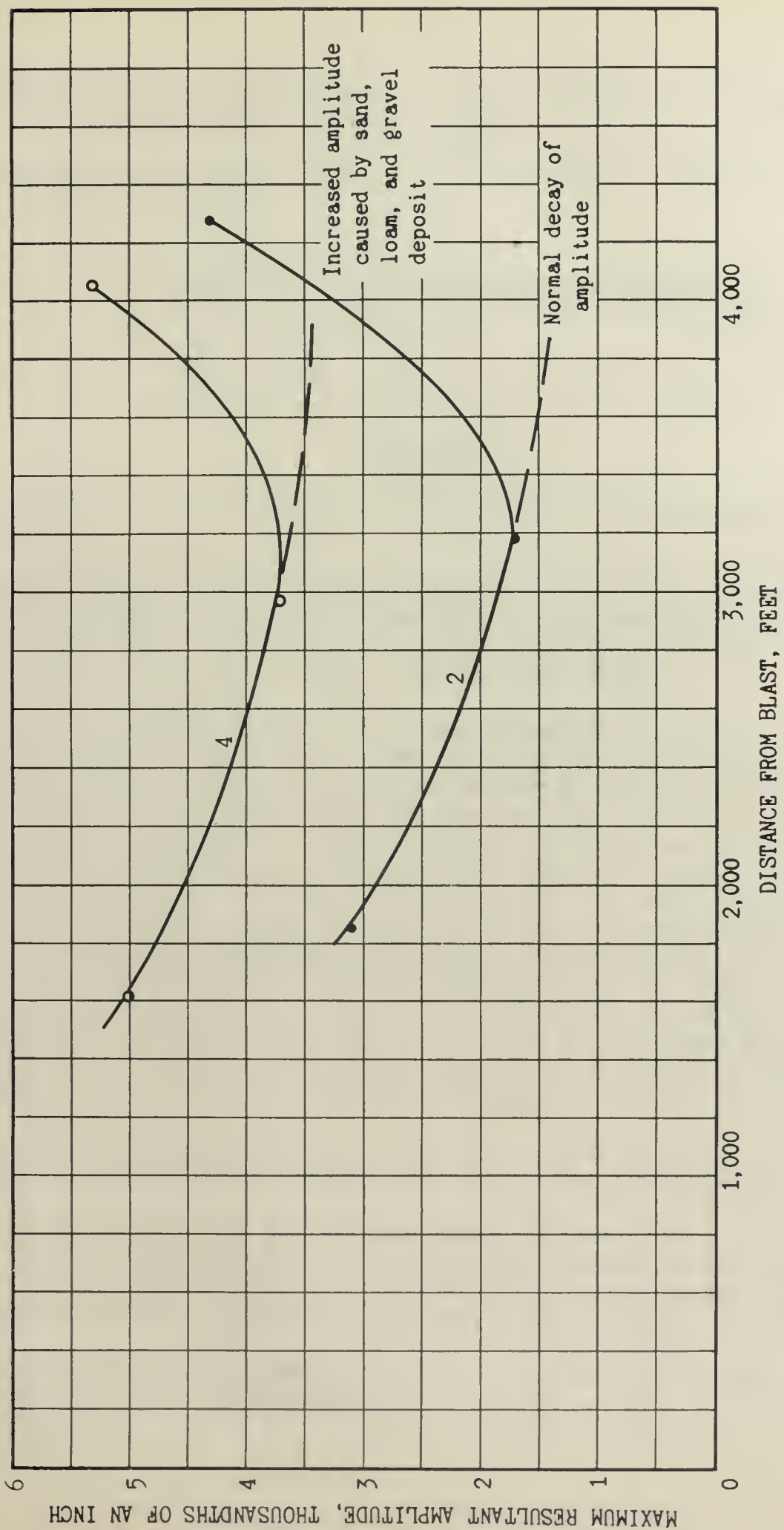


Figure 7.- Amplitude-distance curve, Quarry Q: Curve 2, shot 2, 2,102 pounds; curve 4, shot 4, 4,056 pounds.



Station B was on overburden, a fill of unknown depth, hence the speed of the ground wave could not be calculated.

Assuming a rock speed of 18,500 for this quarry, the overburden depths were computed as follows:

Station	Shot 1	Shot 2	Shot 6	Shot 7	Shot 8
A	24	14	--	--	--
B	27	23	70	88	67
C	57	35	--	--	--

Quarry Q

This quarry lies in heavily bedded dolomite dipping south 65° west at 12° (fig. 6).

Seven well-drill shot tests were made in an attempt to determine the comparative effect of different quantities of the same type of explosives, the same quantity of different explosives, and the difference between blasting on a high bench and low bench with the same charge.

It was originally intended that the seismometer stations should remain identical for all shots. The two nearest stations were constant for all shots. The farthest station, however, was moved closer in after the first shot. The reason for this change was that on the first shot this farthest station gave a record of low frequency and high amplitude characteristic of thick loose overburden. Such a condition was not anticipated at the site of this station. The station however was in a small surface depression and it was thought some local deposit of loose fill in this depression might be causing the unexpected wave shape. Consequently, the station was moved closer to the quarry and placed on a knoll or low ridge. Results from the rest of the shots, however, confirmed that of the first, that both locations of this station were on a large loose gravel fill. Two of the shots that well illustrate the phenomenon of increased amplitude with increased distance because of geologic change are plotted in figure 7.

The quarry face was 100 feet high, but in one section of the quarry the upper portion had been removed leaving a bench 50 feet high.

Five shots were made in the high bench and two in the low bench. The two shots made in the low bench had explosive charges closely comparable with two in the upper bench. Of the five shots in the upper bench, two were comparable as to quantity of explosive, but the type was different.

For convenience the results obtained have been tabulated in three groups to illustrate comparisons.

The seismometer stations were in a line behind the quarry face. Stations A and B were on overburden of undetermined thickness. Station C (two locations) was on loam-sand-gravel of considerable thickness.

Group 1

Shot	Holes	Elevation collar	Explosive, pounds	Station	Horizontal distance, feet	Time first impulse, second	Frequency cycles	Duration seconds
1	1	144	936	A	1,710	0.096	30-50	0.4
		103		B	3,070	.163	30	.7
		67		C	4,150	.233	20-60	.5
							4-10	6.0
2	3	144	2,102	A	1,850	.10	30-50	.5
		103		B	3,180	.167	35	1.0
		67		C	4,260	.237	40-60	.3
							6-10	6.0
3	4	144	3,256	A	1,710	.098	30	.5
		103		B	3,070	.165	30	.7
		49		C	4,560	.248	30-80	.3
							6- 8	6.0
4	4	144	4,056	A	1,610	.11	30-40	.4
		103		B	2,970	.175	25-40	.7
		67		C	4,050	.248	30-80	.3
							5-10	6.0

Shot	Station	Distance, feet	Apparent speed, ft./sec.	Amplitude, inch	Explosive, pounds	Overburden, feet
1	A	1,710	17,900	0.0015	936	25
	B	3,070	18,800	.0018		17
	C	4,150	17,800	.0015		60
2	A	1,850	18,500	.0031	2,102	15
	B	3,180	19,000	.0017		12
	C	4,260	18,000	.0043		56
3	A	1,710	17,500	.0043	3,256	31
	B	3,070	18,600	.0028		23
	C	4,560	18,400	.0027		42
4	A	1,610	14,650	.0050	4,056	82
	B	2,970	17,000	.0037		68
	C	4,050	16,300	.0053		121

It will be noted that the speeds calculated for station B were highest in each case. Study of the dip of the strata reveals that those beds that occur at the juncture of quarry floor and face would outcrop just below station B. Thus, the impulse delivered by the lower portion of the charge in each shot had a direct path through the beds to station B. To reach stations A or C, the wave would have to travel from one bed to another across bedding planes.

The apparent speeds from shot 4 are lower than for the other shots. A crack was observed in the formation behind this shot, which may account for this reduced speed.

Shot 4 was directly on the line of seismometer stations and, consequently, the distances were shortest. The quarry face ran roughly north 20° west and shot 2 was placed north of shot 4, while shots 1 and 3 were adjacent and south of shot 4.

The geologic map of the area showed a line of break in the dolomite striking about north 30° east and cutting the line of seismometer stations about midway between stations B and C. It was demonstrated definitely that this break was a sloping contact between the dolomite and a sand-gravel-clay deposit. The contact dipped south 45° east at about 12° to 15° .

The amplitude at C for shot 3 was less than at A, but for this shot the distance to C was greater. This was the first shot made and following it Station C was drawn in to its new position for the rest of the shots.

All of these holes were shot with a special quarry explosive.

Test Comparing Different Kinds of Explosive

For this test a single hole was shot (shot 5) beside the location of shot 1 and loaded with 900 pounds of 60- and 75-percent gelatin dynamite.

For further comparison, another shot (shot 6) of two holes loaded with 85⁴ pounds of the special explosive was made in the lower bench. Shots 1 and 5 were in the high bench and located south of shot 4. Shot 6 was in the low bench south of shots 1 and 5. The results from this group of holes are tabulated as follows:

Group 2

Shot	Holes	Elevation collar	Explosive, pounds	Station	Horizontal distance, feet	Time, first impulse, sec.	Frequency, cycles	Duration, seconds
1	1	144	936	A	1,710	0.096	30-50	0.4
		103		B	3,070	.163	30	.7
		67		C	4,150	.233	{ 30-50 4-10	{ 6.5 6.0
5	1	144	900	A	1,730	.093	30-40	.4
		103		B	3,070	.163	30	.7
		67		C	4,150	.233	{ 40-80 5-10	{ .3 5.0
6	2	144	854	A	2,170	.118	30-40	.4
		103		B	3,440	.184	30	1.0
		67		C	4,510	.254	{ 30-80 7-10	{ .3 5.0

Shot	Station	Distance, feet	Apparent speed, ft./sec.	Amplitude, inch	Explosive, pounds	Overburden feet
1	A	1,710	17,900	0.0015	936 (Special)	25
	B	3,070	18,800	.0018		17
	C	4,150	17,800	.0015		60
5	A	1,730	18,600	.0016	900 (Dynamite)	13
	B	3,070	18,800	.0017		17
	C	4,150	17,800	.0010		60
6	A	2,170	18,400	.0013	854 (Special)	20
	B	3,440	18,700	.00085		23
	C	4,510	17,750	.0007		69

It will be noted again that for all shots the speed to station B is highest.

It is noticeable that the amplitude at station B for shots 1 and 5 is greater than for station A, although station A was 1,300 feet closer. Reference to the first group of shots shows amplitudes at A greater than at B for all except shot 1.

There are two possible explanations for this. Shots 1 and 5 were both single-hole shots; all others were multiple-hole shots. A delay in firing time of one hole from another may cause complicated wave forms,

accounting for this anomaly. Shots 1 and 5 may have been on the exact point at which the beds in which the charge was placed outcropped under station B. The others may have been above or below this exact point. However, shot 3 was nearly identical in location with shots 1 and 5 and will not check with them in comparative amplitudes for A and B.

Shot 5 was slightly south of shot 3, which was south of shot 1, hence the path angle from shot 5 to station C was still nearer parallelism to the line of contact between dolomite and gravel. Shot 6 was considerably south of 5 and still closer to path parallelism. The amplitudes at station C for both shots are roughly 30 percent less than at A. It is possible that at some point between the locations of shots 3 and 5 the path to station C met the contact plane at such an angle that most of the energy was reflected and less refracted through to the gravel. This would account for the comparatively smaller amplitude at C as the path approached parallelism.

Comparison of the amplitudes for shots 1 and 5 shows little difference, hence it is concluded the vibrational effect is the same for both kinds of explosive.

Test for Difference in Elevation of Explosive Charge

Shot 7 consisted of 3,233 pounds of special explosive charged in a number of holes in the 50-foot bench and located just north of shot 6.

Shots 1 and 3 in the high bench approximate shots 6 and 7 in quantity and kind of explosive. The results from these four shots are presented in the following table:

Group 3

Shot	Holes	Elevation collar	Explosive, pounds	Station	Horizontal distance, ft.	Time first impulse, sec.	Frequency cycles	Duration, seconds
1	1	144	936	A	1,710	0.096	30-50	0.4
		103		B	3,070	.163	30	.7
		67		C	4,150	.233	30-60 4-10	{ .5 6.0
6	2	144	854	A	2,170	.118	30-40	.4
		103		B	3,440	.184	30	1.0
		67		C	4,510	.254	30-80 7-10	{ .3 5.0
3	4	144	3,256	A	1,710	.098	30	.5
		103		B	3,070	.165	30	.7
		49		C	4,560	.248	30-80 6-8	{ .3 6.0
7	9	144	3,233	A	2,110	.115	40	.4
		103		B	3,410	.183	25-40	.9
		67		C	4,470	.253	30-80 5-10	{ .3 6.0

Shot	Station	Distance, feet	Apparent speed, ft./sec.	Amplitude, inch	Explosive, pounds	Overburden, feet
1	A	1,710	17,900	0.0015	936	25
	B	3,070	18,800	.0018		17
	C	4,150	17,800	.0015		60
6	A	2,170	18,400	.0013	854	20
	B	3,440	18,700	.00085		23
	C	4,510	17,750	.0007		69
3	A	1,710	17,500	.0043	3,256	31
	B	3,070	18,600	.0028		23
	C	4,560	18,400	.0027		42
7	A	2,110	18,350	.0033	3,233	20
	B	3,410	18,600	<u>1</u> /.003		25
	C	4,470	17,650	.002		72

1/ For some unknown reason, one instrument did not show a curve on the photograph record.

Again it is noticed that the speed to station B is greatest in all cases.

An amplitude-distance curve shows that if the curve for shot 6 is produced to the 1,710-foot point the value read from the curve is within 10 percent of the amplitude recorded at that distance for shot 1, which corresponds to the difference in explosive charges.

Likewise, a similar curve for shots 3 and 7 will show all amplitudes at distances above or below the curve within the range of readable error.

From these results it is concluded that blasting on the upper or lower bench makes no appreciable difference in the resulting vibration so long as the explosive charges are bottomed at the same elevation.

A study of the apparent speeds of propagation shows the rock speed must have been greater than 19,000 feet per second; therefore, 19,500 is assumed.

On this basis, depths of overburden have been calculated and are given in the tables. A study of these shows that the depths calculated for shot 4 are consistently greater than the others, due to the slower speed recorded for all stations at that shot. Eliminating this shot, the average depth of overburden at A is 21 feet and at B is 20 feet. At C the depth for shot 3 is 42 feet and the average for all other shots, except 4, is 63 feet.

SUMMARIZED RESULTS

The essential features of the preceding quarry tests have been condensed in table 1 for convenient reference.

As discussed in the introduction, a variety of explosive charges, distances, and types of rock were involved in these tests. If the amplitudes are plotted against distance for the explosive charges it is seen that for comparable distances and charges the amplitudes on outcrop are generally one tenth to two thirds that on overburden.

A general formula expressing the relation between amplitude, distance, and explosive charge should indicate the shape of the curve resulting from an amplitude-distance plot and should approximate a curve plotted from the observed data.

The trend of the change in amplitude as distance increases indicates a high degree of initial damping followed by a lower degree. The initial damping is of the order of the inverse cube of the distance. Later damping is roughly inversely proportional to the distance.

The trend of the change in amplitude with change in weight of explosive at constant distance indicates that the amplitude increases with explosive charge at a somewhat slower rate than in direct proportion to the charge. Mathematically the combination of these trends may be expressed in different ways. For example: the result may be obtained over the desired range in distance and explosive charge by introducing a constant that adds to a slowly damped curve and has effect only at the shorter distances, thus, in effect, obtaining a high degree of initial damping followed by a low degree of damping; or the constant can be introduced so that it adds to a highly damped curve but has effect only at the greater distances, thus producing a lesser degree of damping where desired; or the desired condition may be obtained by having the damping exponent change with distance, that is, decrease with the larger distances. These three possible solutions may be written in the following forms:

$$(1) \quad A = KC^S/(d^n + K_1)$$

$$(2) \quad A = (KC^S/d^n) + K_1$$

$$(3) \quad A = KC^S/(K_0 d^m)$$

TABLE 1.

Quarry	Stone	Shot		Explosive			Station			Distance	
		No.	Type 1/	Type 2/	Strength, percent	Weight, pounds	No.	Type 3/	Depth, feet	Hori- zontal, feet	Eleva- tion, feet ^{4/}
A	Dolomite	1	WD	G	60-75-80	15,400	A	Over	38	617	+140
							B	Out		1,365	+183
							C	Over	26	2,075	+286
B	Biotite- gneiss	1	WD	NG	60	11,500	A	Out		640	+ 42
							B	Out		1,100	+ 25
							C	Over	170	1,650	+ 11
C	Limestone	1	WD	NS		250	A	Over	41	451	+ 68
							B	Over	29	684	+ 88
							C	C.fl.	31	740	+ 90
		2	Ch	NS		35	No reading. Amplitude less than .001 inch.				
		3	MC	NS		0.36	A	Over		5/450	---
							B	Over		683	---
		4	MC	NS	30	1.13	A	Over		425	---
							B	Over		660	---
							C	C.fl.		715	---
		5	HD	NS	50	36	A	Over	45	510	---
							B	Over	26	740	---
							C	C.fl.	28	785	---
		6	WD	NS	50	300	A	Over	42	440	+ 68
							B	Over	31	673	+ 88
							C	C.fl.	33	730	+ 90
D	Limestone	1	HD	G	60	232	A	Over	20	443	+ 28
							B	Over	10	641	+ 43
							C	Over	25	882	+ 49
E	Limestone	1	HD	G	40	34	A	Q.fl.		330	0
							B	Q.fl.		400	0
							C	Q.fl.		530	0
		2	HD	G	40	115	A	Q.fl.		5/315	---
							B	Q.fl.		365	---
							C	Q.fl.		515	---
		4	HD	A	40	42	A	Over	44	1,470	+ 29
							B	Over	11	2,460	+ 57
		5	HD	A	40	213	A	Over	44	1,470	+ 9
							B	Over	11	2,460	+ 37
							C	Ho.		2,500	+ 54
		6	HD	A	40	163	A	Over	44	1,470	+ 11
							B	Over	11	2,460	+ 17
							C	Ho.		2,500	+ 34

TABLE 1. - (Cont'd)

Quarry	Time arrival, second	Speed, feet per second		Frequency cycles per second	Duration, second	Amplitude, inch	Air wave	
		Apparent	Rock				Time arrival, second	Speed, feet per second
A	0.04	15,400	22,700	12	0.8	0.058	0.085	7,250
	.06	22,700		20	.5	.010		
	.10	20,750		8	.8	.021		
B	.0525	12,200	26,300	20	.6	.028	.11	5,820
	.07	15,700		20	.6	.016		
	.12	13,750		7	2.0	.054		
C	.04	11,300	17,000±	20-30	.7	.0080	.41 .64 .36 .585 .61 .00015 	

TABLE 1. -- (Cont'd)

Quarry	Stone	Shot		Explosive			Station			Distance	
		No.	Type1/	Type2/	Strength, percent	Weight, pounds	No.	Type3/	Depth, feet	Hori- zontal, feet	Eleva- tion, feet ^{4/}
F	Limestone	1	WD	A	50-60	619	A	Over	44	666	+ 37
							B	Over	15	1,136	+ 38
							C	Q.fl.		1,136	+ 1
		2	WD	A	50-60	543	A	Over	43	520	+ 37
							B	Over	18	1,000	+ 38
							C	Q.fl.		1,020	+ 1
G	Traprock	2	Coyote	G	40	19,500	A	Over	112	1,350	-123
							B	Over	98	1,840	-126
							C	Over	24	2,330	-123
		3	Coyote	G	40	28,200	A	Over	99	2,280	- 86
							B	Over	30	2,750	- 83
							C	Over	18	3,240	- 63
H	Traprock	1	WD	G	60-75	5,100	A	Over	6	1,900	+ 57
							B	Over	18	2,700	- 1
							C	Over	18	4,400	- 19
		2	WD	G	60-75	6,050	A	Over	-	4,510	- 12
							B	Over	-	5,000	- 13
							C	Over	-	5,960	- 35
		3	WD	G	60-75	7,500	A	Over	27	4,010	- 31
							B	Over	90	4,510	- 32
							C	Over	99	5,470	- 44
I	Traprock	1	Coyote	G	40	17,250	A	Over	170	1,760	-110
							B	R.fill	208	1,880	-110
							C	C.fl.	190	1,810	-110
J	Traprock	1	Coyote	G	40	11,100	A	Over	27	2,500	- 55
							B	Over	20	3,000	- 55
							C	Over	84	3,600	- 54
K	Limestone	1	WD	G	40-60-75	13,400	A	Over	123	2,000	-119
							B	Over	117	2,540	-123
							C	Over	42	3,630	-107
		2	WD	G	40-60-75	17,050	A	Over	114	2,090	-119
							B	Over	93	2,630	-123
							C	Over	21	3,720	-107

TABLE 1. - (Cont'd)

Quarry	Time arrival, second	Speed, Feet per second		Frequency cycles per second	Duration, second	Amplitude, inch	Air wave	
		Apparent	Rock				Time arrival, second	Speed feet per second
F	0.05	13,300		20-30	0.3	<u>6</u> /0.005		
	.065	17,500		20-30	.3	<u>6</u> / .0075		
	.06	18,900	18,900	50	.3	.0005		
	.045	11,550		20	.5	.009		
	.065	15,400		20	.5	.008		
	.06	17,000	17,000	50	.2	.0014		
G	.16	8,400	<u>8</u> /11,000	30-80	1.0	.017		
	.20	9,200		&	to	.013		
	.22	10,600		4-10	4.0	.012		
	.24	9,500	<u>8</u> /11,000			<u>6</u> / .012		
	.26	10,600		30-50/4	10/.8/5.0	.012		
	.30	10,800		30/6	<u>10</u> /.2/.4	.005		
H	.125	13,600	17,000±	28	1.0	<u>6</u> / .014		
	.165	15,000		28	1.0	.004		
	.265	15,700		28	1.0	.002		
	<u>9</u> /.05	9,800		28	.8	.0018		
	.06	16,000		28/5	3.0	.0035		
				28/5	3.0	<u>6</u> / .007		
	.245	15,400	17,000±	28	.6	.0036		
	.295	14,600		6	5.0	.004		
	.355	14,800		6	5.0	<u>6</u> / .007		
I	.16	11,000	17,000±	30	.2	.028		
	.18	10,450		3.3	7-8.0	<u>6</u> / .068		
	.17	10,650		3.3	7-8.0	.033		
J	.17	14,700	15,500±	33	.5	.0046		
	.20	15,000		33	.5	.0036		
	.26	13,850		33	.5	.0017		
K	.215	9,300	11,500±	33/5	.4/8.0	.025		
	.26	9,800		33/5	.4/8.0	.016		
	.33	11,000		33/5	.4/8.0	.0082		
	.22	9,500	11,500±	33/5	.4/8.0	.021		
	.26	10,100		33/5	.4/8.0	.014		
	.33	11,300		33/5	.4/8.0	.008		

TABLE 1. - (Cont'd)

Quarry	Stone	Shot		Explosive		Station			Distance		
		No.	Type1/ Type2/	Strength, percent	Weight, pounds	No.	Type3/ Depth, feet	Hori- zontal, feet	Eleva- tion, feet ^{4/}		
L	Gabbro- diorite	1	WD	G	60	1,200	A	Out		1,300	+ 47
							B	Out		1,480	+ 46
							C	Out		1,780	+ 67
		2	WD	G	60	1,200	A	Out		1,230	+ 47
							B	Out		1,400	+ 46
							C	Out		1,700	+ 67
		3	WD	G	60	1,200	A	Out	14	1,270	+ 4
							B	Cr.st.		1,300	- 8
							C	Ho.		1,300	+ 12
		4	WD	G	60	1,200	A	Out	14	1,200	+ 4
							B	Cr.st.		1,240	- 8
							C	Ho.		1,240	+ 12
M	Gabbro- diorite	1	WD	G	60	1,200	A	Out	36	600	+184
							B	Over		1,170	+ 97
							C	Over		28	1,785
		2	WD	G	60	?	A	Out	30	635	+184
							B	Over		1,210	+ 97
							C	Over		21	1,825
		3	WD	G	60	1,150	A	Out	23	665	+184
							B	Over		1,250	+ 97
							C	Over		15	1,865
		4	WD	G	60	1,300	A	Out	33	700	+184
							B	Over		1,280	+ 97
							C	Over		24	1,900
N	Flint	1	HD	G	60-75	348	A	Out		5/330	---
							B	Out		585	---
							C	Out		875	---
		2	HD	G	60-75	337	A	Out		510	---
							B	Out		440	---
							C	Out		370	---
		3	HD	G	60-75	335	A	Out		435	---
							B	Out		490	---
							C	Out		565	---

TABLE 1. - (Cont'd)

Quarry	Time arrival, second	Speed, feet per second		Frequency cycles per second	Duration, second	Amplitude, inch	Air wave		Path distance feet
		Apparent	Rock				Time arrival, seconds	Speed, feet per second	
L	0.085	15,300	16,000±	30-40	0.4	0.0046			
	.09	16,450		30-40	.4	<u>6/</u> .0026			
	.11	16,200		30-40	.4	.0027			
	.08	15,400	16,000±	30-40	.4	.0046			
	.085	16,500		30-40	.4	<u>6/</u> .0035			
	.105	16,200		30-40	.4	.0038			
	.07	18,150	18,000	45	.4	.0021			
	.085	15,300		45/16	.2/1.1	.0047			
	.09	14,450		12-20	2.0	.0071			
	.067	17,900	18,000	20-60	.2	.0028			
	.083	14,950		16	.5	.0077			
	.086	14,400		10-16	1.5	.008			
M	.035	17,950	18,660	50	.25	.017			628
	.075	15,650		16-50	.25	.0076			1,174
	.105	17,000		30-50	.25	<u>6/</u> .002			1,785
	.035	18,900	18,660	50	.25	.008			662
	.075	16,200		16-50	.25	.0032			1,213
	.105	17,400		30-50	.25	<u>6/</u> .0016			1,825
	.035	19,700	18,660	50	.25	.013			690
	.075	16,700		16-50	.25	.0057			1,253
	.105	17,750		30-50	.25	<u>6/</u> .0037			1,865
	.04	18,100	18,660	50	.25	.015			724
	.08	16,050		16-50	.25	.0057			1,284
	.11	17,250		30-50	.25	<u>6/</u> .0028			1,900
N	.0275	12,000		20-50	.2	.01			
	.0475	12,300		20-50	.2	.0027			
	.0675	13,000		20-50	.2	.0022			
	.045	11,350	12,000	30-50	.25	.0033			
	.035	12,600		20-40	.25	.005			
	.03	12,350		30	.25	<u>6/</u> .006			
	.035	12,400	16,000	30-50	.25	.008			
	.04	12,250		25-50	.25	.0057			
	.045	12,550		20-50	.1	<u>6/</u> .006			

TABLE 1. - (Cont'd)

Quarry	Stone	Shot		Explosive			Station			Distance	
		No.	Type1/	Type2/	Strength, percent	Weight, pounds	No.	Type3/	Depth, feet	Hori- zontal, feet	Eleva- tion, feet ^{4/}
N	Flint	4	HD	G	60-75	330	A	Out		5/555	---
							B	Out		455	---
							C	Out		635	---
		5	HD	G	60-75	183	A	Out		615	---
							B	Out		510	---
							C	Out		690	---
		6	HD	G	60-75	355	A	Out		1,125	---
							B	Over	11	1,145	---
							C	C.ap.	12	1,140	---
		7	HD	G	75	1,000	A	C.ap.		4,550	---
O	Traprock						B	Out		4,730	---
							C	Over		4,700	---
		8	HD	G	60-75	203	A	C.ap.	30	1,090	---
							B	Out		1,560	---
							C	Over	3	1,590	---
		9	HD	G	60-75	442	A	Over	33	470	---
							B	Out		730	---
							C	Over	25	740	---
		10	HD	G	60-75	500	A	Over	50	1,330	---
		1	Coyote	G	40	41,800	A	Over	42	3,075	- 80
P	Gabbro- diorite						B	Over	66	2,930	-101
							C	Over	33	2,925	-121
		1	Snake	G	40-60	80	A	Marsh	24	870	+ 2
							B	Over	27	1,500	+ 23
							C	Sandy Over	57	1,780	+118
		2	Snake	G	40-60	100	A	Marsh	14	932	+ 2
							B	Over	23	1,525	+ 23
							C	Sandy Over	35	1,820	+118
		3	Snake	G	60	100	A	Over		2,130	+125
		4	Snake	G	60	100	A	Over		4,160	+167
		5	Snake	G	60	100	A	Over		7,440	- 19

TABLE 1. - (Cont'd)

Quarry	Time arrival, second	Speed, feet per second		Frequency cycles per second	Duration, second	Amplitude, inch	Air wave	
		Apparent	Rock				Time arrival, seconds	Speed, feet per second
N	0.035	15,850		30-50	0.2	0.002		
	.03	15,150	12,000	30-50	.2	?		
	.04	15,900		30-50	.2	.0021		
	.0375	16,400	to	30-50	.2	.0018		
	.0325	15,700		30-50	.2	.0026		
	.04	17,250		30-50	.2	.0012		
			16,000					
	.075	15,000		30-50	.25	.00085		
	.08	14,300		20	.8	.00245		
	.08	14,300		10-15	1.5	.00375		
						.00016		
						.00008		
						.00011		
	.08	13,600		15-50	1.0	.00225		
	.10	15,600	15,600	40-50	.5	.0007		
	.103	15,450		20-50	.5	.0012		
	.04	11,750		25-30	.5	.02		
	.045	16,200	16,200	40	.2	.0025		
	.054	13,700		20-40	.5	.0095		
	.105	12,650	15,000±	14	1.0	.0031		
O	.20	15,400	11/17,000	5	1.2	.033		
	.22	13,300		5	1.2	.020		
	.23	12,700	8/11,000	5	1.2	<u>6/</u> .043		
P	.055	15,800	18,500±	80-20	.4-.8	.0013		
	.09	16,650		80-30	.2-1.0	.0008		
	.115	15,500		80-30	.2-.8	.0006		
	.055	16,950		80-20	.4-.8	.0012		
	.09	16,950		80-30	.2-1.0	.00055		
	.11	16,550		80-30	.2-.8	.00035		
				60-20	.7	<u>6/</u> .0008		
				100-40	.7	.00026		
				50-40	.7	<u>6/</u> .000055		

TABLE 1. - (Cont'd)

Quarry	Stone	Shot		Explosive			Station			Distance	
		No.	Type ₁ /	Type ₂ /	Strength, percent	Weight, pounds	No.	Type ₃ /	Depth, feet	Hori- zontal, feet	Eleva- tion ₄ / feet
P	Gabbro- diorite	6	Snake	G	60	80	A	Rockpile	.	135	+ 39
							B	Over	70	495	+ 25
		7	Snake	G	60	100	A	Rockpile		180	+ 39
							B	Over	88	566	+ 25
		8	Snake	G	60	100	A	Out		210	+ 15
							B	Over	67	422	+ 25
Q	Dolomite	1	WD	N		936	A	Over	25	1,710	+144
							B	Over	17	3,070	+103
							C	Over	60	4,150	+ 67
		2	WD	N		2,102	A	Over	15	1,850	+144
							B	Over	12	3,180	+103
							C	Over	56	4,260	+ 67
		3	WD	N		3,256	A	Over	31	1,710	+144
							B	Over	23	3,070	+103
							C	Over	42	4,560	+ 49
		4	WD	N		4,056	A	Over	82	1,610	+144
							B	Over	68	2,970	+103
							C	Over	121	4,050	+ 67
		5	WD	G	60-75	900	A	Over	13	1,730	+144
							B	Over	17	3,070	+103
							C	Over	60	4,150	+ 67
		6	WD	N		854	A	Over	20	2,170	+144
							B	Over	23	3,440	+103
							C	Over	69	4,510	+ 67
		7	WD	N		3,233	A	Over	20	2,110	+144
							B	Over	25	3,410	+103
							C	Over	72	4,470	+ 67

1/ WD = Well-drill, Ch = churn-drill, MC = mud cap, HD = hammer-drill, Coyote = tunnel shot, Snake = horizontal holes under face.

2/ G = Gelatin, NG = nitroglycerin, NS = nitro-starch, A = ammonia, N = special.

3/ Over = overburden, out = outcrop, C.fl. = concrete floor, Q.fl. = quarry floor, Ho. = house floor, R. fill = rock fill, Cr. st. = crushed stone, C.ap. = concrete apron.

4/ Elevation in feet above (+) or below (-) bottom of holes.

5/ Path distance.

6/ Estimated.

TABLE 1. - (Cont'd)

Quarry	Time arrival, second	Speed, feet per second		Frequency cycles per second	Duration, second	Amplitude, inch	Air wave	
		Apparent	Rock				Time arrival, second	Speed, feet per second
P	0.05	9,900		40	0.25	<u>6</u> /0.0012	0.03	4,500
	.06	9,400		40	.25	<u>6</u> / .0017	.055	3,270
	.045	9,400		30	.75	.0025	.07	3,000
Q	.096	17,800	19,500±	30-50	.4	.0015		
	.163	18,800		30	.7	.0018		
	.233	17,800		30-60	.5	.0015		
				4-10	6.0			
	.10	18,500		30-50	.5	.0031		
	.167	19,000		35	1.0	.0017		
	.237	18,000		40-60	.3	.0043		
				6-10	6.0			
	.098	17,500		30	.5	.0043		
	.165	18,600		30	.7	.0028		
	.248	18,400		40-80	.3	.0027		
				6-8	6.0			
	.11	14,650		30-40	.4	.0050		
	.175	17,000		25-40	.7	.0037		
	.248	16,300		30-80	.3	.0053		
				5-10	6.0			
	.093	18,600		30-40	.4	.0016		
	.163	18,800		30	.7	.0017		
	.233	17,800		40-80	.3	.0010		
				5-10	5.0			
	.118	18,400		30-40	.4	.0013		
	.184	18,700		30	1.0	.00085		
	.254	17,750		30-80	.3	.0007		
				7-10	5.0			
	.115	18,350		40	.4	.0033		
	.183	18,600		25-40	.9	<u>6</u> / .003		
	.253	17,650		30-80	.3	.002		
				5-10	6.0			

7/ Dolomite.8/ Sandstone.9/ Time interval between stations.10/ 30-50 cycles for 0.8 second, 4 cycles for 5.0 seconds.11/ Traprock.

In these equations:

A = Amplitude of vibration.

K = A constant depending upon the units used.

C = Weight of explosive.

s = A constant indicating the rate of increase of amplitude with increasing charge.

d = Distance of shot from recording point.

n = A constant indicating the rate of damping with distance. This rate of damping predominates over the greater distances in the case of equation (1), and over the shorter distances in the case of equation (2).

K_1 = A constant introduced so as to have great effect at the shorter distances in the case of equation (1); at the greater distances in the case of equation (2). It is seen that when " d " is very small in equation (1), K_1 dominates the denominator and determines the magnitude of the result; however, when " d " is very large, K_1 is negligible. In equation (2), if " d " is small, the quotient is large and K_1 is negligible; however, if " d " is large, the quotient is correspondingly small and K_1 dominates the result.

K_0 = A constant depending upon the units chosen.

m = An exponent that decreases in value with increasing distance; that is, the rate of damping decreases at the greater distances.

Different solutions of the three forms shown in equation (1), (2), and (3) were examined. A modified form of equation (2) was found to be the most satisfactory formula. This modified form is:

$$(4)a \quad A = (KC^s/e^{nd}) + C^sK_1, \quad \text{or}$$

$$b \quad A = KC^s(e^{-nd} + K_1).$$

This equation (4) shows a logarithmic decrease with distance. The constant " n " is called the "damping coefficient." In equation (2) the constant, K_1 , is not multiplied by the charge, but in equation (4) it will be noted that the charge affects the constant K_1 as well as the term e^{-nd} ; however, the constant serves the same purpose as already noted for equation (2).

The numerical values for the constants were found to be:

$$s = 2/3 = 0.67$$

$$K = 7/10000 = 0.0007$$

$$K_1 = 1/70 = 0.014$$

$$n = 0.00143$$

$$e = 2.72 = \text{base of napierian logarithms.}$$

These values are used when:

C = explosive charge in pounds,

d = distance in feet,

A = maximum resultant amplitude in inches.

The resultant amplitude is obtained by taking the magnitude of the vector sum of the three components of displacement at any given instant. This is the same as taking the square root of the sum of the squares of the magnitudes of the three components of displacement at any given instant. The maximum resultant amplitude is found by obtaining the resultant amplitudes over the period of time desired and selecting the largest of these. In any case, the resultant amplitude at a given time is not more than the square root of three (1.73) times the largest single component at that given time, nor smaller than the largest single component. The former case is illustrated when all three components are equal; the latter case when two components are zero. The resultant amplitude expresses the magnitude of the vibration from point of rest in any direction. The maximum resultant amplitude is the greatest displacement or movement from point of rest occurring during the vibration regardless of what direction that greatest displacement happens to take.

The final suggested form of the equation, which is equation (4) with the constants evaluated is:^{7/}

$$(5) \text{ a } A = 0.0007C^{2/3}(e^{-.00143d} + .014)$$

$$\text{or b } A = \frac{C^{2/3}}{10^4} (7e^{-.00143d} + 1/10)$$

$$\text{or c } A = \frac{C^{2/3}}{100} (.07e^{-.00143d} + .001)$$

Substituting for distance and explosive charge in formula (5) gives the maximum resultant amplitude expected. The amplitudes were computed for

^{7/} This formula is tentative and is indicative only of generalized conditions. Because of local variables, it cannot be applied accurately to any single problem.

10, 100, 1,000, 5,000, and 10,000 pounds of explosive charge. For each of these weights distances of 100, 300, 500, 1,000, 1,500, 2,000, 3,000, and 6,000 feet were considered. The results compare favorably with the summarized data, although in some cases individual differences between the computed and the observed amplitudes for a particular shot and distance are great. This is shown in table 2, in which a number of shots were selected at random and the amplitudes computed from formula (5).

TABLE 2. - Amplitudes

Quarry	Explosive charge, pounds	Distance, feet	Maximum resultant amplitudes, inch	
			Observed	Computed ^{1/}
A	15,400	1,365	0.01	0.07
B	11,500	640	.028	.15
C	250	684	.0054	.01
	36	710	.00082	.0029
E	34	400	.002	.0043
	163	2,460	.0028	.0009
F	543	520	.009	.023
K	13,400	2,000	.025	.028
L	1,200	1,300	.0046	.014
Q	3,256	3,070	.0028	.0041
P	100	7,440	.00006	.0002
	100	4,160	.00026	.00026
N	348	585	.0027	.015

^{1/} Differences between observed amplitudes and those computed by formula are due to the impossibility of accurately calculating the variables inherent in each local test.

An exact formula, which would give better agreement between the predicted and observed amplitudes, would include all the variables that affect the amplitude; for example, the physical characteristics of the underlying rock, the position of the recording point with relation to the bedding planes, etc. The value of the variables would have to be determined for each particular quarry before the formula could be used. The formula becomes more complex as more variables are introduced.

Even though formula (5) neglects all variables except weight of explosive charge and horizontal distance from shot to recording point, in a few cases the computed amplitude is close to the observed amplitude; in most cases the ratio of computed to observed is 3:1 (or 1:3) or better. In a few cases the ratio is greater than 3:1, but in none of the cases examined did the ratio exceed 10:1. The formula was designed to give satisfactory results between 100 and 6,000 feet, and from explosive charges ranging from 10 and 15,000 pounds. Above 15,000 pounds data are scarce, and therefore the applicability of the formula is questionable; beyond 6,000 feet the error in the formula becomes appreciable, owing principally to the constant K_1 .

Because of geologic differences, a general mathematical formula can be only approximate and the variation between the observed amplitudes and those computed from a mathematical formula will be as much as, if not greater than, the variation between amplitudes or overburden and outcrop at a particular distance and charge.

A study of the observed frequencies of vibration indicates that there were no outstanding differences that could be attributed to the variation in type of rock. On the outcrops the frequencies ranged from 20 to 80 cycles per second but in most cases were between 20 and 50. Some frequencies on overburden also fell in this group, but in a number of observations they were between 4 and 10 cycles per second. No frequency observed on outcrop was below 20 cycles, and in every case where conditions were comparable the frequency on the outcrop was greater than the frequency on overburden.

Table 3 shows the observed and estimated speeds of propagation of the seismic waves through the various rocks. The estimated speeds were calculated where direct observations on outcrop were not available.

TABLE 3. - Rock speeds, feet per second

Rock	Observed			Estimated		
	Minimum	Average	Maximum	Minimum	Average	Maximum
Biotite-gneiss...	-	26,300	-	-	-	-
Dolomite.....	-	22,700	-	14,700	15,900	19,500
Flint.....	12,000	15,200	16,000	-	15,000	-
Gabbro-diorite...	16,000	17,800	18,700	-	18,500	-
Limestone.....	8,000	13,200	18,900	11,500	16,300	24,000
Sandstone.....	-	-	-	-	11,000	-
Traprock.....	-	-	-	15,500	16,700	17,000

The results seem to indicate that the speeds are higher in the denser rocks, but exact correlation cannot be made without tests designed specifically to obtain the speeds of propagation.

Whereas the speed of propagation of the seismic wave is of primary importance to the geophysicist, it is of secondary interest only in this problem. Nevertheless, in order that all available data be made a matter of record, rock speeds have been measured wherever possible and calculated or assumed in other cases. Rock speed becomes important in this problem when it is not possible to find outcrops on which to set the seismometer stations and they must be established on overburden, the depth of which is not known. At times, the depth can be estimated closely from well records, road cuts, or nearby outcrops. In other cases no information is available and the speed in both rock and overburden must be assumed.

In a few instances speeds in overburden have been recorded direct where the distances have been short enough to insure against reflected wave paths. These have varied greatly, ranging from 500 feet per second in loose loam and decayed vegetable matter to 6,000 feet or more in clay. In this report, in all but a few instances the speed in overburden has been assumed arbitrarily as 3,000 feet per second.

Where outcrops have been available for seismometer settings, rock speeds have been calculated direct from the time of blast, time of receipt of first impulse at the seismometer, and distance between the blast and seismometer.

Where the depth of overburden could be estimated closely, that depth has been used with an assumed speed of 3,000 feet per second and the remaining time calculated as time of travel through the rock.

Where depth of overburden is unknown, a rock speed has been assumed from other records where similar geologic conditions and rock structure existed and the remainder of the time interval used to calculate overburden depth. Figure 4 shows this type of calculation graphically.

In table 4 appear the observed maximum resultant amplitudes, the frequencies associated with these amplitudes, and other factors computed from them. As already mentioned, the maximum resultant amplitude is larger than; but not more than the square root of three (1.73) times; the largest component, and therefore is of the same order of magnitude as the largest component. Where the frequency is not constant, the highest frequency observed is tabulated; thus, the acceleration derived from these values is presumably the maximum acceleration present in the vibration. The solution for acceleration using equation (6) assumes that the vibration took the form of a sine wave.

The following equation is commonly used to calculate acceleration:

$$(6) \quad a = Aw^2 \sin.wt.$$

in which

a = acceleration,

A = amplitude,

$w = 2\pi f$ where f = frequency of vibration,

t = time at which acceleration is desired.

This formula assumes that the vibration is sinusoidal or, in other words, took the form of a sine wave. Some of the recorded waves approximated this form closely; others diverged from it materially, in which case a separate formula is necessary for each and would require long and intricate mathematical analysis. Therefore, equation (6) was used to compute the accelerations in table 4 with the reservation that the results are not exactly representative of the true conditions, if the waves recorded are not sinusoidal.

TABLE 4.

Quarry	Shot	Station	Maximum resultant amplitude A, inch	Frequency f cycles/sec	A(2 π f) ² accelera- tion mm/sec ²	Mercalli scale modified, intensity	A(2 π f) ³ change in accelera- tion cm/sec ³	$\frac{W(2\pi fa)^2}{2g}$ kinetic energy inch-lb.	Seismometer location
A	1	C	0.021	8	1,350	10	6,800	0.00144	Overburden.
B	1	A	.028	20	11,240	12+	141,000	.0160	Outcrop.
C	1	B	.0054	30	4,900	12	92,000	.00134	Overburden.
	1	C	.0029	30	2,600	11	49,300	.00037	Building.
	5	B	.00082	50	2,050	10	64,500	.00009	Overburden.
E	2	B	.014	25	8,800	12	13,800	.0063	Outcrop.
H	2	A	.0036	28	2,800	11	49,600	.00052	Overburden.
J	1	A	.009	33	9,900	12	205,000	.0045	do.
L	1	A	.0046	40	7,400	12	186,000	.0017	Outcrop.
N	1	A	.010	50	25,000	12+	788,000	.013	do.
L	3	A	.0021	50	5,200	12+	164,000	.00051	do.
P	1	C	.0006	30	530	9	10,000	.000018	Overburden.
N	10	A	.0031	20	1,250	10	15,700	.00021	do.
Q	5	B	.0017	30	1,530	10	29,000	.00014	do.
	4	A	.005	40	8,030	12	202,000	.00205	do.
E	5	C	.0004	20	158	7	1,980	.000004	Building.
L	-	A	.0007	40	1,120	10	28,200	.000041	do.

As the maximum value for $\sin.wt.$ is 1, the table is based on the value of Aw^2 . In addition, the value of Aw^3 is also tabulated and is the rate of change of the acceleration with respect to time, as was suggested for an index of destructiveness in a previous publication^{8/}. The acceleration is stated in terms of millimeters per second per second in order that a direct comparison may be made with the Modified Mercalli Intensity Scale of 1931 by Wood and Neumann. It is seen that the index is 12 for many observations, and this index value is defined thus: "Damage total. Waves seen on ground surface. Lines of sight and level distorted. Objects thrown upward into the air." It is obvious that the comparison reduces to an absurdity.

In table 4 also appear the kinetic energies calculated for each vibration. The kinetic energy tabulated is that for a 1-pound weight ($W = 1$). To obtain the value for any other weight, multiply the indicated kinetic energy by the weight in pounds. The kinetic energies derived are comparable with those obtained by Professor E. H. Rockwell^{9/}.

In view of the tests made and the calculations presented in table 4, the authors are of the opinion that the Mercalli Scale is not applicable to the frequencies and amplitudes of disturbances caused by quarry blasting.

Field tests to date do not supply sufficient evidence on which to calculate an index of destruction.

Various observations were made on the speed of propagation of the air disturbance caused by blasting. Table 5 shows that no speed below 1,000 feet per second was observed, but several records indicate speeds in the order of 5,000 feet per second.

TABLE 5. - Air blast

Quarry	Station	Speed, ft/sec	Distance, feet	Charge, pounds
A	A	7,250	617	15,400.00
B	A	5,820	640	11,500.00
C	A	1,100	450	.36
C	B	1,070	683	.36
C	A	1,180	425	1.13
C	B	1,130	660	1.13
E	C	1,020	530	34.00
E	C	1,040	2,500	213.00
E	C	1,040	2,500	163.00
P	A	4,500	135	80.00
P	A	3,270	180	100.00
P	A	3,000	210	100.00

^{8/} Lee, F. W., Thoenen, J. R., and Windes, S. L., Earth Vibrations Caused By Quarry Blasting: Rept. of Investigations 3319, Bureau of Mines, Nov. 1936, 19 pp.

^{9/} Rockwell, E. H., Vibrations Caused by Blasting and Their Effect on Structures: Hercules Powder Co., Wilmington, Del., 1934, 69 pp.

In the immediate vicinity of the blast, the compressional wave or air disturbance travels at high speed, which is greater than the speed of sound and which is attributed to actual displacement of the air rather than a vibratory wave passing through it and leaving it relatively undisturbed, as is the condition when a less violent shock occurs. The speed decreases as the distance from the shot increases and finally becomes the speed of sound. The decrease of speed with distance may be observed best in the data from quarry P, in which the amount of charge was nearly constant. No attempt was made to distinguish between the air disturbance and the audible or sound wave. No attempt was made to record any abnormally low speeds that might be present as indicated by Partlo and Service^{10/}. For this reason, average speeds only are shown and no computations are made of the instantaneous speeds.

In summarizing vibrations in buildings it is noted that the excitation may be of two types - vibration produced through coupling with the ground and vibration produced through impact of the air on the structure. In quarry C it is observed that the vibration resulting from the latter is comparable to that resulting from the former, even though the measurement was made on the basement (ground level) floor. At quarry E the air vibration caused a floor motion greater than that resulting from the ground motion. The predominance of the horizontal vibrations resulting from the air blast indicates that the observed motion was not floor motion alone, which probably would be largely vertical, but represented movement of the whole structure horizontally. At quarry L the vibration observed in the building resulting from the ground vibrations is comparable to that on the crushed stone yard just outside. From these data it is seen that any study of building vibration from blasting must consider both possible sources - air blast and ground vibration. The two can easily be recorded separately because the speed of the air disturbance is always considerably slower than the speed of the vibration through the underlying rock.

^{10/} Partlo, F. L., and Service, J. H., Instantaneous Speeds in Air of Explosion Reports at Short Distances from the Source: Physics, vol. 6, no. 1, Jan. 1935, p. 1.

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REPORT OF INVESTIGATIONS

HARDENING OF MUD SHEATHS IN CONTACT WITH OIL,
AND A SUGGESTED METHOD FOR MINIMIZING
THEIR SEALING EFFECT IN OIL WELLS



BY

C. P. BOWIE

REPORT OF INVESTIGATIONS

UNITED STATES DEPARTMENT OF THE INTERIOR - BUREAU OF MINES

HARDENING OF MUD SHEATHS IN CONTACT WITH OIL, AND A
SUGGESTED METHOD FOR MINIMIZING THEIR SEALING EFFECT IN OIL WELLS^{1/}

By C. P. Bowie^{2/}

INTRODUCTION

For years many oil producers have believed that often low-pressure oil- and gas-bearing strata have been overlooked in fields where the rotary system of drilling has been used. Collom^{3/} gave as a reason for this condition that because the mud fluid was thick, the oil or gas stratum was plastered and sealed as the tools passed through it and no oil showed at the surface. Although this observation was made 15 years ago, it may often still be true notwithstanding modern developments in coring and electrical logging.

Operators have believed, also, that mud fluid does not solidify behind the casing. Lewis and McMurray^{4/} stated: "All the evidence obtainable from practical experience points to the fact that the mud fluid properly mixed and employed does not solidify behind the casing, even in deep wells and after long periods."

This statement may be true where the mud fluid is in contact with water-bearing formations, but the author believes that other conditions hold where the mud fluid is dehydrated, as it can be by contact with crude oil.

In the fall of 1934, while experimenting with mud fluid in the San Francisco laboratory of the Bureau of Mines, the author found that drilling mud apparently solidifies after being immersed in crude oil for a relatively short time.

^{1/} The Bureau of Mines will welcome reprinting of this paper, provided the following footnote acknowledgment is used: "Reprinted from Bureau of Mines Report of Investigations No. 3354."

^{2/} Supervising engineer, Bureau of Mines, San Francisco, Calif.

^{3/} Collom, R. E., Prospecting and Testing for Oil and Gas: Bull. 201, Bureau of Mines, 1922, p. 149.

^{4/} Lewis, J. O., and McMurray, W. F.; The Use of Mud-Laden Fluid in Oil and Gas Wells: Bull. 134, Bureau of Mines, 1916, p. 15.

Laboratory observations made in connection with this work indicate that where wells have been drilled through oil-producing formations of low differential pressure, a sheath of mud cake has been formed between the casing and the surface of the formation, which after a time becomes as hard and as impervious to the passage of oil as the cap rock that holds the oil and gas in place in the structure. Actually, the clay in the drilling fluid, which later became a mud cake behind the casing, was a pulverized part of that cap rock or some similar material.

After "drilling in" oil wells with mud fluid, operators often are disappointed with the relatively small production of the well as compared with the favorable oil "showings" in cores. The mud sheath that has been deposited on the wall of the well opposite the oil- and gas-bearing formation may account for the smaller production than anticipated, as this sheath will inhibit the flow of oil and gas from the formation into the well. For this reason oil frequently is used as a circulating fluid after the well has been drilled to within a few feet of an oil stratum.

However, certain problems arise incident to the use of oil as a drilling fluid. Its velocity and viscosity during circulation should be great enough to remove the cuttings to the surface. A mud fluid depends largely on its colloidal properties for removing the cuttings. The oil used for circulation may be viscous enough to carry the cuttings to the surface before an oil stratum has been encountered. However, the viscosity will be reduced if a zone is penetrated that produces an oil of lower viscosity. This reduction may be sufficient to prevent satisfactory removal of the cuttings.

Jensen^{5/} in discussing "drilling in" with oil mentions a well completed in California during 1936 where about 18,000 barrels of oil were used as a drilling fluid. Circulation to the surface was not re-established in drilling the last 800 feet of the hole. The drill cuttings apparently went into cavities in the formations.

In discussing the controlled-pressure method of drilling where oil or a mixture of oil and gas is used as a circulating medium instead of mud and where the flow from the well is controlled by the use of special equipment while drilling is in progress, Ross^{6/} states: "Although the controlled-pressure method of completing wells has been found applicable to fields where the underground strata are relatively hard and capable of standing up in open hole, its application to areas where the hole penetrates great thicknesses of unconsolidated and loosely-cemented formations is still questionable."

5/ Jensen, Joseph, Recent Developments Related to Petroleum Engineering: Oil and Gas Jour., vol. 35, no. 21, Oct. 8, 1936. pp. 44, 48. Also, Trans. Am. Inst. Min. and Met. Eng., vol. 123, Petroleum Development and Technology, 1937, pp. 63-68.

6/ Ross, John S., Review of Controlled-Pressure Drilling Method: The Petroleum Engineer, vol. 7, no. 13, September 1936, pp. 116, 118, 120-121.

Various other methods have been used to complete oil wells; these include reaming, swabbing, shooting, and chemical treatment. Any method will have some limitations. The author believes that in some applications the method suggested in this paper may have certain advantages not inherent in the other methods.

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The assistance and cooperation of Frank F. Smith and Elmer J. Willey, of San Francisco, in the laboratory during the early experimental work are especially to be acknowledged.

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MUD HARDENING - OBSERVATIONS AND THEORY

To verify the apparent finding that mud fluid solidifies when in contact with crude oil, clay from Coalinga, Calif., like that used in Kettleman Hills oil field, was mixed with enough water to form a plaster that would adhere to the walls of a cylinder about 4 inches in diameter by 6 inches in length. The cylinder was made from 20-mesh stamp-mill battery screen. The mud was plastered on the inside wall to a thickness of $3/8$ inch. The cylinder with the mud cake then was submerged in a jar filled with 39.2° A.P.I. crude oil from the Belridge (Calif.) field. A watch glass was placed on top of the jar, thereby retarding evaporation of the lighter portions of the oil. As the time of submergence increased, the A.P.I. gravity of the oil decreased. This arrangement showed the effect of oil of varying gravity on the mud cake.

After being immersed in the oil for 10 days, the mud cake showed signs of solidifying; and after being in the oil 22 months, it attained a hardness about equal to that of a brittle shale. When the cake was dropped from the laboratory bench to the floor the sound of the impact was equivalent to that made by an equal amount of shale, and the mud cake was so solid that it did not fracture. Moreover, when the cake was placed in a test tube of water it would not disperse as would a clay suspension when shaken.

Other muds, ranging in consistency from a soft plaster to a 78.3-pound-per-cubic-foot drilling mud, were mixed from Coalinga (Calif.) clay and Mojave (Calif.) clay. Each of these muds was poured into a flat-bottom flask of 1/2 to 3 liters capacity. The bottom of each flask was covered to a depth of 1/2 inch with the mud sample. These containers then were filled to within 1 inch of their tops with 39.5° A.P.I. Kettleman (Calif.) crude oil and sealed.

The stiffer muds showed signs of solidifying in 10 days. In 10 months, even when the flasks were turned on their sides so that the disks of mud stood on edge, no mud flow took place. The cakes appeared to have the hardness of plaster of paris that has set. The sample of 78.3-pound drilling mud was placed on the bottom of a 3-liter flask. After standing over night the mud was covered with 1/8 inch of "free" water. Most of this water was not visible after a few days had passed.

Mud cakes 1 inch in diameter by 3/8 inch in thickness, prepared by adding just enough water to wet the dry clay and then molding under a pressure of 100 pounds per square inch, after being immersed in oil for 30 days, had the approximate hardness of a similar cake dried in air.

In the ceramics industry much research has been performed and numerous theories developed to explain the plastic properties and solidification of clays. The theories will not be discussed in this paper, but it is believed generally that clay mixed with the proper amount of water and allowed to dry "sets" or becomes hard due to the action of water on the colloidal parts of the clay and the subsequent removal of the water.

According to Wilson^{7/}, a concept associated with the plasticity of clay is: "On drying or elimination of the water a coherent brittle mass is produced, stronger than the plastic clay."

Clays mixed with water to form a drilling mud apparently harden when immersed in oil because enough of the water is removed by the oil to cause the clay to "set". Such clays (mud cakes) therefore are akin to air-dried clays, except that air-dried clays are capable of being returned to a clay suspension on the addition of water while oil-dried clays are not, or certainly not readily so. Lockwood^{8/} found that the water content of 36.5° B. oil increased 6 percent after being in contact with water for 13 days.

^{7/} Wilson, Hewitt., *Ceramics - Clay Technology*: McGraw-Hill Book Co., Inc., New York, 1927, p. 58.

^{8/} Lockwood, Robinson P., *Role of Cap Rock in Oil Accumulation*: Bull., Am. Assoc. Petrol. Geol., vol. 17, no. 6, June 1933, p. 725.

Possibly the fact that plastic clays harden when in contact with crude oil has an important bearing on the accumulation of oil in underground strata.

SUGGESTED TREATMENT OF MUD FLUID

Difficulties may arise due to the solidification of mud behind the casing opposite an oil- and gas-bearing formation or due to the adherence of a mud sheath to the wall of a producing formation. To correct these conditions the author suggests that a suitable material soluble in crude oil and insoluble in water might be added to the mud fluid before the potentially productive zones are penetrated. This treatment should render the sheath permeable to oil. Of course, the material added must not interfere with the normal functions of the mud and should remain in suspension during circulation.

If several oil- and gas-bearing strata are penetrated during the course of drilling the well, it might be advisable to use a treated mud fluid throughout. Then, after "plugging back" it would be possible to obtain production from the upper low-pressure zones after the deeper, more prolific zones have been exhausted, because the sheath opposite the oil-bearing strata would be permeable to oil.

Materials Considered for Mud Treatment

Apparently various substances have required properties, but all of them may not have practical value. Asphalts, naturally occurring and manufactured, and metallic soap, such as calcium stearate, are soluble or partly soluble in hydrocarbons and virtually insoluble in water; however, the limited experimental work the author has done with these materials yielded few favorable results. The gum-forming latex, of plant life, such as crude rubber, rosin, and chicle, seem to be the most desirable, especially those that have the property of swelling before going into solution. The swelling shatters the mud cake, opening channels in it, and when the added material has gone into solution a free passage is provided for the oil and gas.

Crude rubber obtained in the form known to the trade as "smoked sheet" has given the best results in the laboratory experimental work. Reclaimed rubber, such as that derived from old automobile tires, will not be suitable as it has been vulcanized. The vulcanizing process very materially reduces the swelling properties as well as the solubility. Para crude and Malay crude rubber have been used with about equal success. These grades are dark amber in color and have a specific gravity of 0.90 to 0.95. Apparently they are readily soluble in hydrocarbons and insoluble in water; both distilled water and that having a relatively high mineral content, such as an oil-field subsurface water. They swell slightly when in contact with water after a long period of time. When in contact with gasoline for a few hours they swell enormously (13 volumes or more) and then pass slowly into a pseudo solution.

According to Webber^{9/}, no definite temperature can be given for the melting point of rubber. With rising temperature it grows softer and stickier, eventually approaching a fluid state at between 170° and 180° C. (338° and 356° F.)

The best results with rubber have been obtained when using strips about 1/16 inch in diameter and 1/2 inch in length. The shredded rubber was mixed with the drilling mud in proportions ranging from 5 to 10 per cent by weight of the dry clay and rubber.

The specific gravity of a 78-pound-per-cubic-foot- drilling fluid made from Mojave (Calif.) clay is 1.25. The specific gravity of the drill cuttings carried to the surface by such a mud is approximately 2.6. Laboratory experiments to be described show that where the crude rubber is thoroughly mixed with a suitable drilling mud the specific gravity of the rubber is near enough to the specific gravity of the drilling fluid to insure uniform distribution of the rubber particles throughout the fluid during circulation. Rubber is tenacious and probably would retain its size and shape after long periods of circulation in mud fluids containing abrasive drill cuttings. The rubber should separate in the ditches because of its lower specific gravity and more rapid rate of flow than the drilling mud. If it does not separate readily, probably the use of a properly adjusted screen will solve the problem. The cuttings should separate from the treated mud in the usual manner as they do from untreated mud.

Some apparently favorable results in the laboratory in rendering the mud cake permeable to oil have been obtained by using common rosin in granular form. This material is derived from the distillation of turpentine oil from crude turpentine. It is usually dark amber, has a specific gravity of about 1.1, and a melting point of 100° to 140° C. (212° to 284° F.) It is insoluble in water and readily soluble in hydrocarbons; however, it lacks the swelling properties afforded by rubber.

Chicle is another substance from which some favorable results have been obtained. This material is used extensively in the manufacture of chewing gum. It is the latex product of one and perhaps more^{10/} species of the genus *Achras* (Sapotaceae). This plant grows wild and also is cultivated extensively in various countries in Central America and the tropical part of South America. Chicle is only partly soluble in water but readily soluble in hydrocarbons.

9/ Webber, Lothar E., *The Chemistry of Rubber Manufacture*: Charles Griffin and Co., Ltd., 1926, p. 57.

10/ Laan, Joseph W. Van der, *Production of Gutta-Percha, Balata, Chicle, and Allied Gums*: U. S. Bureau of Foreign and Domestic Commerce Trade Promotion Series 41, 1927, p. 49.

P. L. Becker, vice president, American Chicle Co., has informed the author that the specific gravity of chicle when dried is about 0.975; before drying it is slightly greater than 1.0. It has no definite melting point but softens rather rapidly within the range of 83° to 87° C. (181° to 188° F.). Probably 85° C. (185° F.) could be taken as its melting point. It is a very complex mixture of resins, gutta (rubber), carbohydrates, mineral matter, and a small amount of proteids. There is no single solvent that will dissolve it completely. The resins and gutta are soluble in hydrocarbons and the carbohydrates in water. There is usually 3 to 5 percent mineral matter insoluble in all ordinary solvent. The approximate composition of chicle is as follows:

	<u>Percent</u>
Gutta.....	12.5
Resins.....	62.5
Carbohydrates.....	15.0
Ash.....	10.0

As has been stated, asphalt also seems to have possibilities. Grades A and B of the California product in granular form have been tried. Grade A, according to A.S.T.M. standard test methods, has a penetration of 0 to 5, melts at 277° F., and is 53 percent soluble in 86° A.P.I. petroleum ether. Grade B has a penetration of 4 to 10, melts at 159° F., and is 65 percent soluble in 86° A.P.I. petroleum ether. This material lacks swelling qualities.

Metallic soaps also are to be considered. Zinc stearate and calcium stearate have been tried in both powdered and granular form. Like asphalt, these materials lack swelling properties. So far, laboratory tests with them have given no favorable results.

EXPERIMENTAL PROCEDURE AND APPARATUS

Several types of laboratory apparatus have been designed to investigate the practicability of treating drilling muds with oil-soluble, water-insoluble materials. Some of the earlier types may be of interest to investigators confronted with similar problems. They will be described, although the results obtained with them were qualitative rather than quantitative.

Hollow-Glass Cylinder Apparatus

To show the relative effect of crude oil on treated or untreated mud plasters, several hollow-glass cylinders 2 inches outside diameter and 10 inches in length were fitted with a cork in one end and attached to ring stands in an upright position to form a series of 500-cc. containers. Each cylinder then was fitted with a mandrel made from conduit pipe of such diameter that an annular space of $\frac{3}{8}$ inch remained between the outer wall of the conduit pipe and the inner wall of the glass cylinder. To facilitate the removal of the mandrel, it was wrapped with one layer of oiled paper before it was inserted into the glass cylinder.

The annular space in each cylinder then was filled with a treated or an untreated 83-pound mud made from Coalinga (Calif.) clay. The treated muds contained, respectively, 10 percent of granulated rosin, 5 percent of granulated rosin, and 6.4 percent of Para crude rubber by weight of treat-clay mixture. The rubber was cut into angular pieces having a largest dimension of 1/8 inch.

After the mud had dried a sufficient length of time (48 hours), the mandrel was removed carefully and the inner space filled with 38.4° A.P.I. Burbank (Okla.) crude oil. The upper end of each cylinder was covered with a watch glass, and fresh oil was added from time to time as the lighter constituents evaporated.

Table 1 shows the results of this experimental work:

TABLE 1. - Untreated and treated 83-pound mud made from Coalinga (Calif.) clay. Cakes 3/8 inch thick set in 2-inch O.D. hollow glass cylinders in contact with 38.4° A.P.I. Burbank (Okla.) crude oil.

Cylinder	Treatment	Lapsed time (days)	Condition of Mud Plaster
A	Untreated	1	A few small cracks near top of tube.
		59	Little change.
		101	Hair-line cracks developing in various parts of cylinder.
		189	Cracks wider and more numerous. No very large cracks.
B	10 percent granulated rosin.	56	Many small cracks. Brown smudges of oil on glass.
		98	Smudges increasing and cracks widening.
		188	Many cracks and smudges dividing cake into smaller sections than 5 percent rosin mix but not so small as in rubber mix.
C	5 percent granulated rosin.	1	Extensive cracks near top and half way down cylinder.
		59	Numerous large cracks to within 3 inches of bottom of cylinder.
		101	Brown blotches developing on sides of cylinder.
		191	Many large cracks but not so numerous as in 10 percent rosin mix or rubber mix.
D	6.4 percent Para rubber	1	Extensive cracks near top of cylinder.
		59	Many small cracks to within 3 inches of bottom of cylinder.
		101	Cracks wider - extending to bottom of cylinder. Many brown blotches on glass.
		191	Innumerable cracks and blotches. Cake badly disintegrated.

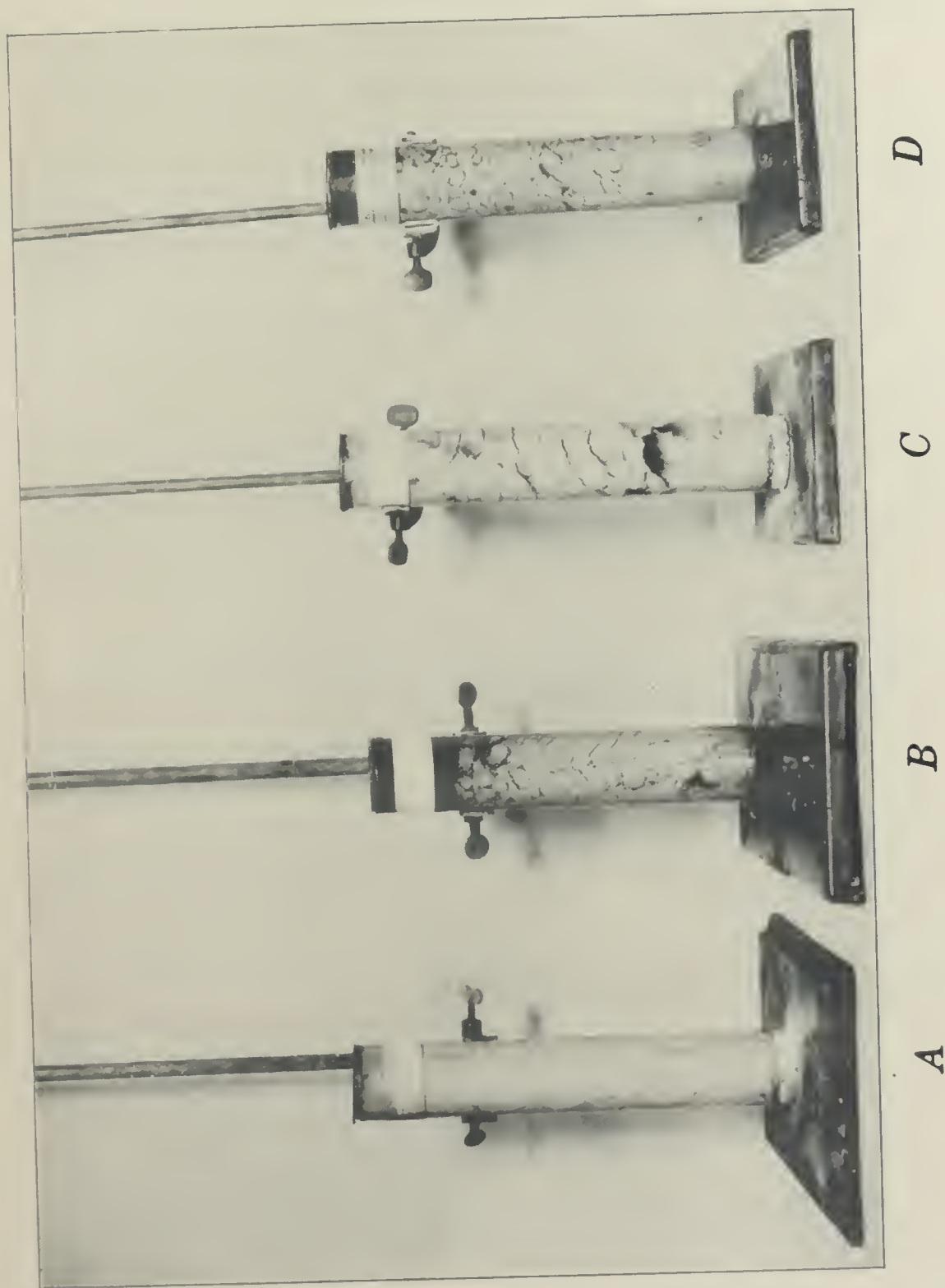


Figure 1.—Condition of mud cakes after being in contact with crude oil for about 6½ months: A, Mud untreated; B, mud treated with 10 percent granulated resin; C, mud treated with 5 percent granulated resin; D, mud treated with 6.4 percent Para crude rubber.

Figure 1 shows the condition of the cylinders after the mud cake had been in contact with the oil for 6-1/2 months.

Although no quantitative results were obtained, this experiment showed that the treated muds cracked more extensively than the untreated mud and that the mud treated with rubber showed the greatest amount of disintegration.

Experiments in Well-Type Apparatus

Another type of apparatus with which considerable work was carried on consisted of a miniature well made from unglazed drain tile 4 inches inside diameter and 2 feet in length. This tile was coated inside and out with a solution of silicate of soda, except for a length of about 6 inches at one end. The lower uncoated end was fitted with a cork, the upper end with a cork in the center of which a hole was drilled large enough to admit a 1-inch O.D. cylinder. One end of a glass tube, 1-inch O.D. and 3 feet in length, then was inserted in the upper cork. A framework made from 3/8-inch-square pump packing then was constructed on the outer surface of the uncoated portion of the tile. The framework was divided into four sections, each to be filled with mud sheath having an area of about 4 by 6 inches and a thickness of 3/8 inch. When the hollow-tile cylinder and glass tubing were filled with oil, a 10-foot oil column resulted. The maximum pressure due to this head was exerted on the inside wall of the tile at its lower end opposite the framework.

An untreated mud cake mixed to about the consistency of mortar used in masonry construction was plastered into one of the frames. The other frames were filled with mud cakes treated with varying quantities of oil-soluble materials and mixed as nearly as possible to the same consistency as the untreated cake.

The tile was placed in a glass jar about 10 inches in diameter by 12 inches in height made by removing the upper part of a 5-gallon water bottle. A wood cover was constructed for the jar. It was made in two sections and cut to fit snugly about the outer circumference of the drain tile. The drain tile and the glass tube then were filled with 39.0° A.P.I. North Belridge (Calif.) crude oil. The oil, due to the pressure head, percolated slowly through the side of the tile near the bottom and came in contact with the treated and untreated mud cakes.

Several experiments were performed with this type of apparatus. The mud plasters were prepared by mixing clay and water with granulated rosin, powdered rosin, granulated chicle, powdered grade A asphalt, powdered zinc stearate, granulated calcium stearate, and chopped crude rubber, respectively. These experiments also showed that the treated mud cakes cracked more rapidly and extensively than the untreated cakes. The cakes treated with rubber developed the most cracks. The effect of the oil on the treated mud in this type of apparatus could be observed more readily than in the hollow-glass cylinders because the mud sheath was on the outside of the drain tile in full view of the observer.

Well Type Using Sandstone for Porous Medium

The essential parts of another type of apparatus, built to simulate field conditions more closely, are shown diagrammatically in figure 2. It was of steel construction and consisted of an 8-inch piece of 6-inch pipe, the ends of which were faced parallel. By means of gaskets, a tight joint could be made between the ends of this pipe and two 6-inch standard flanges. The upper flange was drilled and tapped at the center for 1-1/2-inch pipe. The flanges were held in place by four 1/2-inch bolts, as shown by the figure.

The experimental work done with this type of apparatus was begun by using unglazed terra-cotta drain tile for the porous medium. This material was discarded because it was too impervious to permit passage of oil under the low differential pressures used in the test.

A hollow cylinder cut from San Jose (Calif.) sandstone then was used to represent the oil-bearing formation. The ends of the cylinder were faced parallel. A 5-inch blank flange was placed at the bottom of the cylinder. Bolts passing through the lower 8-inch flange, as shown in the diagram, could be tightened against the 5-inch flange, thereby making tight joints between the ends of the cylinder, the upper 6-inch flange, and the 5-inch flange. A 1/8-inch drain pipe led from the 5-inch flange through a stuffing box in the lower 8-inch flange.

San Jose sandstone is a relatively porous rock used as a building stone in some parts of California. In San Luis Obispo County, Calif., oil is found in it. However, the particular sandstone used in the experiments was not oil-bearing.

To operate this apparatus, a reservoir made from 1-1/2-inch pipe about 4 feet long was attached to the 1-1/2-inch nipple in the upper 6-inch flange. Mud fluid was poured into the reservoir and the sandstone cylinder until they were filled. A mud cake was "set" in the cylinder by admitting air under a pressure of 70 pounds per square inch at the top of the reservoir. Water removed from the mud fluid while the cake was being built was forced through the walls of the sandstone cylinder and then drained through the stuffing box, which had been loosened. The pressure was held for 48 hours.

At the end of the pressure period the fittings were removed from the nipple in the upper flange. This nipple was used as a guide for a wood auger. A hole was drilled through the mud to the bottom of the sandstone cylinder. This formed a well, the walls of which were plastered with a consolidated mud cake 1/2 inch thick.

Oil was introduced into the annular space between the sandstone cylinder and the 6-inch line pipe. By exerting the necessary pressure on the incoming oil it could be forced through the sandstone cylinder against the mud lining.

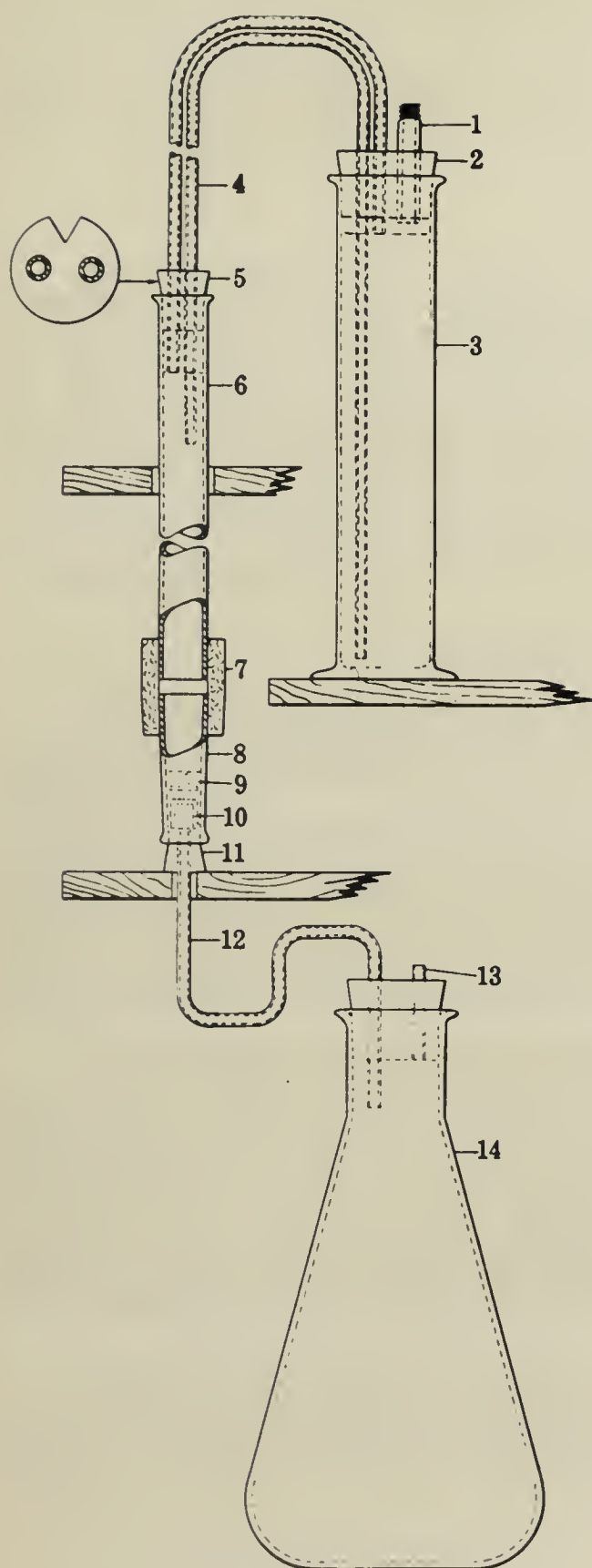


Figure 3.—Glass-tube type.

LEGEND

- 1 -- Filling pipe
- 2 -- Rubber stopper
- 3 -- Oil reservoir
- 4 -- Glass tubing
- 5 -- Cork, showing enlarged top view
- 6 -- Glass tubing
- 7 -- Oil resistant hose
- 8 -- Tapered glass tubing
- 9 -- Mud cake
- 10 -- Brass support
- 11 -- Cork
- 12 -- Glass tubing
- 13 -- Vent
- 14 -- Receiving flask

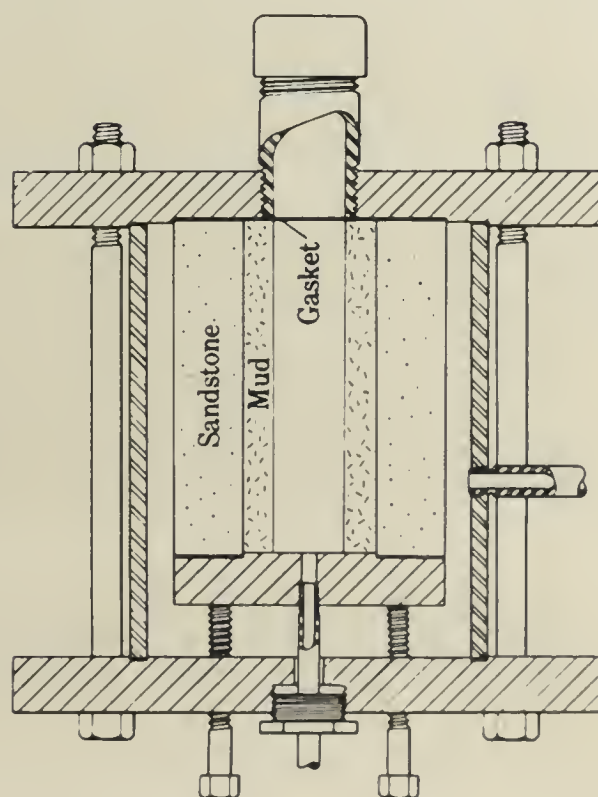


Figure 2.—Well type using sandstone for porous medium.

To prevent cracking of the cake by drying, a sponge enclosed in a small wire cage after being soaked with water was placed in the well. A cap was screwed on the 1-1/2-inch nipple. The sponge was resoaked at least once every 24 hours. The experiments were carried on at room temperature. A battery consisting of six units of this type of apparatus was constructed, and a number of experiments were conducted.

Oil Used

The oil used at the outset of these experiments was 38.5° A.P.I. Kettleman (Calif.) crude. Gumlike material formed and clogged the pores in the terra-cotta pipe and sandstone so rapidly that it was soon impossible to force oil into the well under a pressure of 80 pounds per square inch. To overcome this difficulty, a blend of refined oils containing 40 percent gasoline, 20 percent kerosene, and 40 percent S.A.E. 30 lubricating oil was used. This blend had characteristics (insofar as gasoline and burning oil content were concerned) that resembled closely crude oil from Kettleman Hills, Calif., having a gravity of 38.5° A.P.I. It was used in all subsequent experiments.

Influence of Permeability

It became apparent from the outset that the permeability of the porous media would affect materially the results obtained from the use of this type of apparatus, as it was necessary to force water through a medium in one direction while the mud cake was being formed, and later to force oil in the opposite direction. For this reason the relative permeability was determined for each sandstone cylinder before the mud fluid was introduced. The relative permeability was also determined for each of the cylinders after the experiment was completed and the mud cakes had been thoroughly cleaned from the surface of the sandstone by washing with water.

The assembly was such that oil could be supplied to all of the cylinders at the same time under any desired pressure from 5 pounds to 80 pounds per square inch. No. 1 well contained the untreated mud cake. The relative permeabilities of all the wells, therefore, were determined with respect to well no. 1. This was done by determining the amount of oil per hour passing through the various cylinders under a given pressure for the same length of time and dividing the amount passing through each of the other wells by that passing through well no. 1.

After the mud cake has been deposited in a well and the surplus mud drilled out, the annular space outside the sandstone cylinder was filled with oil, which was held at the desired pressure until the trial was completed.

Table 2 gives the condensed results of a typical experiment using various differential oil pressures in which the mud cake in no. 1 well was untreated, that in no. 2 treated with 20 percent chopped Para crude rubber (largest dimension about 3/16-inch) that had been "milled" for about 1 hour, in no 3, 20 percent granular rosin, in no. 4, 20 percent powdered zinc stearate, in no. 5, 20 percent granulated chicle, and in no 6, 20 percent granulated grade B asphalt. Percentages were determined by weight of treat material plus dry clay.

$$\frac{(\text{Weight of treat material})}{(\text{Weight of treat material plus dry clay})} \times 100 = \text{percent}.$$

The mud fluid was made from Mojave (Calif.) clay ground to pass 35-mesh. The experiment was conducted at room temperature (about 70° F.).

Column 7, the corrected yield, was obtained by multiplying the relative permeability of the well at the particular pressure (column 5) by the yield of well no. 1 (column 6) at that same pressure. The result is intended to indicate the amount the well would have produced had the porous medium in it been identical with the porous medium used in well no. 1 and the mud cake in it been untreated. Column 8 is obtained by subtracting column 7 from column 4, and column 9 by dividing column 8 by column 6 and multiplying by 100. Column 14 gives the average of column 9 and column 13.

All of the wells lined with treated muds showed decided increases in production over well no. 1 when using the original relative permeabilities determined for the various sandstone cylinders before water had come into contact with them. The greatest increase occurred when the differential pressure was lowest - 6 pounds per square inch. In well no. 3, containing mud treated with 20 percent granular rosin, this increase was 1,810 percent. In well no. 6, containing mud treated with 20 percent granulated grade B asphalt, the increase in production under the 6-pound differential pressure was 1,210 percent.

The percent increase in production calculated, using relative permeabilities obtained after the mud cakes had been removed, were in all cases much less. In well no. 3 for differential pressures in excess of 10 pounds there was a decrease in yield (indicated by minus signs in the table), and in well no. 4 there was a decrease for every differential pressure.

It will be seen that there is a large discrepancy between many of the corresponding relative permeabilities before and after the experiment. For example, in well no. 3 under the 6-pound differential pressure the variation is 5.08 (442 percent). Probably the variance was continuous at all times in all of the wells.

TABLE 2. - Percent increase in production of blended oil at various pressure differentials
due to treatment of mud fluid made from Mojave red clay

Well no. 1 - Untreated													
1	2	3	4	5	6	7	8	9	10	11	12	13	14
Date, 1936	Pres- sure, lb. per sq. in.	Hours	Yield per hour (cc.)	Rela- tive permea- bility (before)	Yield, well no. 1 (cc. per hour)	Correct- ed yield (cc. per hour)	Yield in- crease due to treat (cc. per hour)	Percent in- crease	Rela- tive permea- bility (after)	Correct- ed yield (cc. per hour)	Yield in- crease due to treat (cc. per hour)	Percent in- crease	Average percent increase
Mar. 21	6	24	375										
" 27	10	24	1.88										
Apr. 4	14	24	5.38										
" 8	20	24	9.25										
" 10	26	24	16.1										
" 11	30	24	19.9										
Well no. 2 - Treated with 20 percent chopped Para crude rubber													
Mar. 21	6	24	1.53	0.744	0.375	0.279	1.30	347.	1.54	0.578	1.00	267	307
" 27	10	24	4.29	.775	1.88	1.46	2.83	151.	1.53	2.88	1.41	75.0	113
Apr. 4	14	24	3.46	.759	5.38	4.08	4.38	81.4	1.43	7.69	.77	14.	48.
" 8	20	24	15.3	.751	9.25	6.95	8.3	90.	1.43	13.2	2.1	23.	57.
" 10	26	24	25.9	.740	16.1	11.9	14.0	87.0	1.38	22.2	3.7	23.	55.
" 11	30	24	31.1	.729	19.9	14.5	15.6	83.4	1.41	28.1	3.0	15.	49.
Well no. 3 - Treated with 20 percent granular rosin													
Mar. 21	6	24	7.21	1.15	0.375	0.431	6.78	1810	6.23	2.34	4.87	1300.	1550
" 27	10	24	13.8	1.16	1.88	2.18	11.6	617	5.40	10.2	3.6	190	410
Apr. 4	14	24	22.3	1.17	5.38	6.29	16.0	297	4.97	26.7	-4.4	-82.	108
" 8	20	24	36.2	1.16	9.25	10.7	25.5	276	4.49	41.5	-5.3	-57.	110
" 10	26	24	53.8	1.16	16.1	18.7	35.1	218	4.18	67.3	-13.5	-85.9	67.0
" 11	30	24	60.8	1.16	19.9	23.1	37.7	189	3.76	74.8	-14.0	-70.4	60.0

TABLE 2. - Percent increase in production of blended oil at various pressure differentials due to treatment of mud fluid made from Mojave red clay (Cont'd)

Well no. 4 - Treated with 20 percent powdered zinc stearate													
1	2	3	4	5	6	7	8	9	10	11	12	13	14
Date, 1936	Pressure, lb. per sq. in.	Hours	Yield per hour (cc.)	Relative permeability (before)	Yield, well no. 1 (cc. per hour)	Corrected yield (cc. per hour)	Yield increase due to treat (cc. per hour)	Percent increase	Relative permeability (after)	Corrected yield (cc. per hour)	Yield increase due to treat (cc. per hour)	Percent increase	Average percent increase
Mar. 21	6	24	0.313	0.484	0.375	0.182	0.131	34.9	0.912	0.342	-0.029	-7.7	13.6
" 27	10	24	1.48	.545	1.88	1.02	.46	24.	.852	1.62	.14	-7.4	8.5
Apr. 4	14	24	4.08	.515	5.38	2.77	1.31	24.3	.798	4.29	.21	-3.9	10.2
" 8	20	24	6.58	.527	9.25	4.87	1.71	18.5	.791	7.32	.74	-8.0	5.3
" 10	26	24	10.5	.543	16.1	8.74	1.80	11.0	.770	12.4	1.9	-12.	.5
" 11	30	24	12.6	.555	19.3	11.0	1.60	8.0	.719	14.3	1.7	-8.5	.3
Well no. 5 - Treated with 20 percent granular chicle													
Mar. 21	6	24	1.46	1.02	0.375	0.383	1.08	288	1.81	0.679	+0.78	+210	+250
" 27	10	24	4.25	1.09	1.88	2.05	2.20	117	2.10	3.95	+ .30	+ 16	66.5
Apr. 4	14	24	10.3	.995	5.38	5.35	4.9	91	2.09	11.2	- .9	- 20	36.0
" 8	20	24	20.4	.984	9.25	9.10	11.3	122	2.22	20.5	- .1	- 1	60.5
" 10	26	24	35.0	.978	16.1	15.7	19.3	120	2.16	34.8	+ .2	+ 1	60.5
" 11	30	24	43.3	.978	19.9	19.5	23.8	120	2.16	43.0	+ .3	+ 2	61.0
Well no. 6 - Treated with 20 percent granular Grade 3 asphalt													
Mar. 21	6	24	5.15	1.56	0.375	0.585	4.54	1210	2.17	0.814	+4.32	+1150	1180
" 27	10	24	11.0	1.65	1.88	3.10	7.9	420	2.14	4.02	-7.0	+370	395
Apr. 4	14	24	14.9	1.51	5.38	8.12	6.8	130	2.16	11.6	+3.3	+ 61	95.0
" 8	20	24	25.8	1.49	9.25	13.8	12.0	130	2.21	20.4	+5.4	+ 58	94.0
" 10	26	24	42.5	1.48	16.1	23.8	18.7	116	2.14	34.5	+5.0	+ 50	83.0
" 11	30	24	49.9	1.47	19.9	29.3	20.6	104	2.09	41.6	+8.3	+ 42	73.0

The author believes that the results obtained with this type of apparatus were too inconsistent to be of much value. However, the construction of the apparatus made it possible to observe the action of the oil (which presumably was exuding from the sandstone at all points on its inner face) on the mud cakes when the wells were producing under any of the pressures used.

The mud cake in well no. 1 (untreated) showed virtually no change under the pressure range. The production appeared to come only from the bottom of the sandstone. Apparently the oil found a passage between the mud cake and the face of the sandstone. Under a pressure of 6 pounds per square inch the well showed no production until the third day.

Well no. 2 (20 percent crude rubber) after being under a pressure of 6 pounds for 5 days showed bulging and sloughing of the lining near the bottom. Production began on the third day. On the sixth day, still under the same pressure, most of the fluid produced consisted of a thick viscous liquid apparently composed of oil and partly dissolved rubber. The pressure was held at 6 pounds for 24 days. During this period the rubber expanded so greatly that it was necessary to clean out the bottom of the well several times as material sloughed from the sides had filled it to a depth of 1 inch or more and completely clogged the drain pipe. At the end of this period oil and partly dissolved rubber was exuding from the mud lining in many places. The run was continued for 19 days more. Although pressures were increased successively until 30 pounds per square inch was attained, there was no appreciable change in the appearance of the mud lining.

Well no. 3 (20 percent granular rosin) under 6 pounds pressure showed an appreciable production on the third day. On the fifth day the oil produced appeared to have a considerable amount of rosin in partial solution. On the sixteenth day, still under a 6-pound pressure, the walls had a roughened appearance and had turned darker in color. Oil production appeared to be from the bottom of the sandstone.

Well no. 4 (20 percent powdered zinc stearate) produced an appreciable amount on the third day under a pressure of 6 pounds. On the fourteenth day the sides of the well had a roughened appearance and on the sixteenth day slight sloughing showed near the bottom. From that time until the end of the run there was no marked change in the appearance of the well. Production appeared to be from the bottom only.

Well no. 5 (20 percent granulated chicle) began to produce on the third day under 6 pounds pressure. The oil produced was dark in color and probably contained some chicle in solution. On the seventh day (under a pressure of 6 pounds) the walls near the bottom began to slough. On the eighth day the sloughing had increased by such an amount that it was necessary to clean out the well. Slight sloughing appeared again on the twenty-first day at 10 pounds pressure. Production appeared to be from the bottom only.

Well no. 6 (20 percent granulated grade B asphalt) produced an appreciable amount on the second day under 6 pounds pressure. The oil was very dark, probably due to the asphalt in solution. On the thirtieth day, when the pressure had reached 20 pounds, the oil produced was still quite dark in color. The walls of the well were almost black and had a shrunken appearance. Throughout the experiment the walls remained smooth and oil production appeared to be quite uniformly distributed over the surface of the lining indicating that the rate of flow through the mud cake was uniform.

Glass-Tube Type

In order to obviate discrepancies due to the variation of permeability of a porous medium, a type of apparatus was designed, one unit of which is shown diagrammatically in figure 3. This experimental equipment consisted of a 1-inch O.D. glass tube 7 feet 6 inches long, mounted vertically on a wooden frame. A tapered glass tube ($1/16$ -inch per inch) 3 inches long (having a greatest diameter equal to that of the long tube) was attached to the bottom of the long tube by a length of oil-resistant hose. A battery of these units was built.

A mud cake $3/8$ -inch thick and containing just enough water to wet it thoroughly was set in the short glass tube under a pressure of 100 pounds per square inch. After the short tube had been connected to the long tube, a brass disk $1/32$ -inch thick, perforated with seven $3/16$ -inch holes, (see Fig. 4, B) was placed $1/8$ inch below the mud cake. This disk was supported on a brass frame resting on a cork inserted in the bottom of the tube. The length of the frame could be shortened slightly by pushing it into the cork. As the mud cake dried it shrank, but there was little or no oil leakage between the cake and the walls of the tube because the tube was tapered and the cake could move downward.

After a cake was set and the glass tube placed in the frame the tube was filled with oil. The oil was deep enough to produce a pressure of 3 pounds per square inch on the cake. The oil head in the tube was kept constant by a siphon feed from a reservoir. This reservoir consisted of a hollow glass cylinder 2 inches in diameter by 12 inches in length placed on top of the wood frame. As oil passed through the mud cake and ran out the bottom of the tube it was replaced immediately by fresh oil from the reservoir unless the cake had failed to such an extent that the volume of outgoing oil exceeded the capacity of the siphon feed.

This automatic refilling apparatus was satisfactory for the experimental work, although there were times when the tube would overflow slightly. This was due to changes in barometric pressure and temperature or to small air leaks developing in the reservoir connections. To operate the siphon, it was necessary, of course, to vent the tube to the atmosphere. This permitted evaporation of some of the oil in the tube, usually from 7 to 10 cc. per day.

Table 3 gives the results of a number of experiments conducted with this type of apparatus. It will be seen that cakes treated with rubber failed in from 5 to 21 days. One cake treated with a mixture of 2-1/2 percent rubber and 2-1/2 percent granulated rosin failed in 16 days, and a cake treated with 15 percent rosin in large pieces (up to 1/8 inch) failed in 25 days. The rate of flow through this cake was rapid immediately after its failure but very slow thereafter. This might indicate that when the rosin had gone into solution the space it had occupied became clogged with mud. Cakes treated with other materials held as well as those receiving no treatment.

TABLE 3. - Comparison of untreated and treated mud cakes^{1/}

Tube	Treatment	Lapsed time before failure (days)	Lapsed time since cake was set (days)	Condition of cake at time of failure	Remarks
1	None	58		Hardened and unbroken	Slow flow around cake.
2	None		126		Cake still holding.
3	None		72		Do.
4	3 percent rubber	21		Swollen and disintegrated	Rapid flow of oil through cake after failure.
5	do.	13		do.	Do.
6	5 percent rubber	17		do.	Do.
7	do.	5		do.	Do.
8	7 percent rubber	5		do.	Do.
9	10 percent rubber	7		do.	Do.
11	15 percent rubber	6		do.	Do.
12	2-1/2 percent rubber 2-1/2 percent granulated rosin	16		do.	Do.
13	15 percent granulated rosin		128		Cake Still holding.
14	do.		128		Do.

TABLE 3. - Comparison of untreated and treated mud cakes^{1/} (Cont'd)

Tube	Treatment	Lapsed time before failure (days)	Lapsed time since cake was set (days)	Condition of cake at time of failure	Remarks
15	15 percent rosin in large pieces (to 1/8 in.)	25		Fractured	High rate of flow through cake for one day, followed by very slow rate.
16	15 percent granulated chicle		128		Cake still holding.
17	do.		128		Do.
18	5 percent granulated Grade A asphalt		127		Do.
19	10 percent granulated grade A asphalt		56		Do.
20	15 percent granulated calcium stearate		127		Do.
21	10 percent granulated zinc stearate		127		Do.
22	10 percent powdered hexachloroethane		104		Do.

^{1/} Table prepared February 3, 1937.

Figure 4, A, shows a mud cake treated with 7 percent Para crude rubber cut into shreds about 1/16 inch in diameter by 3/8 inch in length that failed in 5 days. B shows the cake in the tube behind the perforated brass disk after failure. C shows the cake after being removed from the tube and gives a better idea of its badly fractured condition.

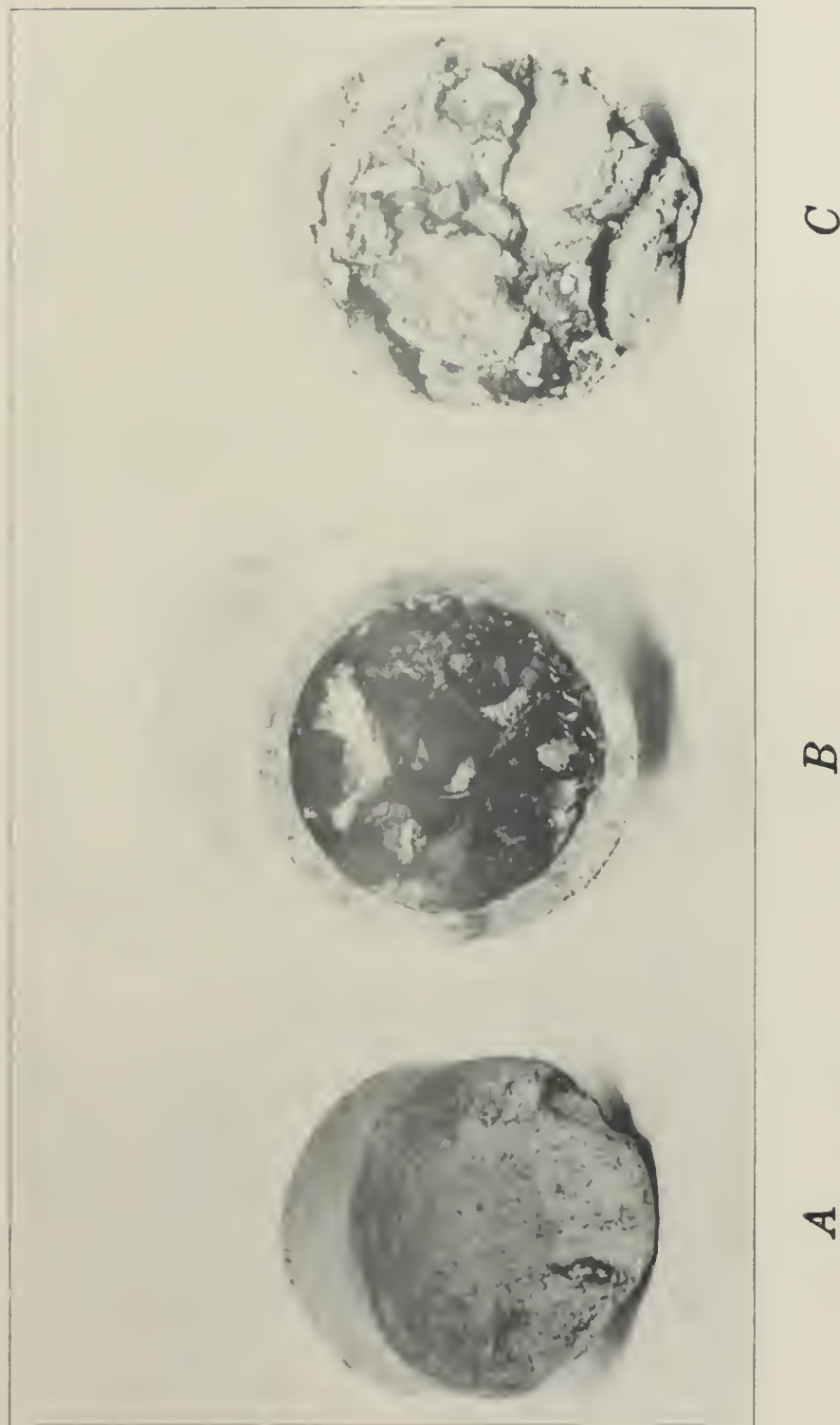
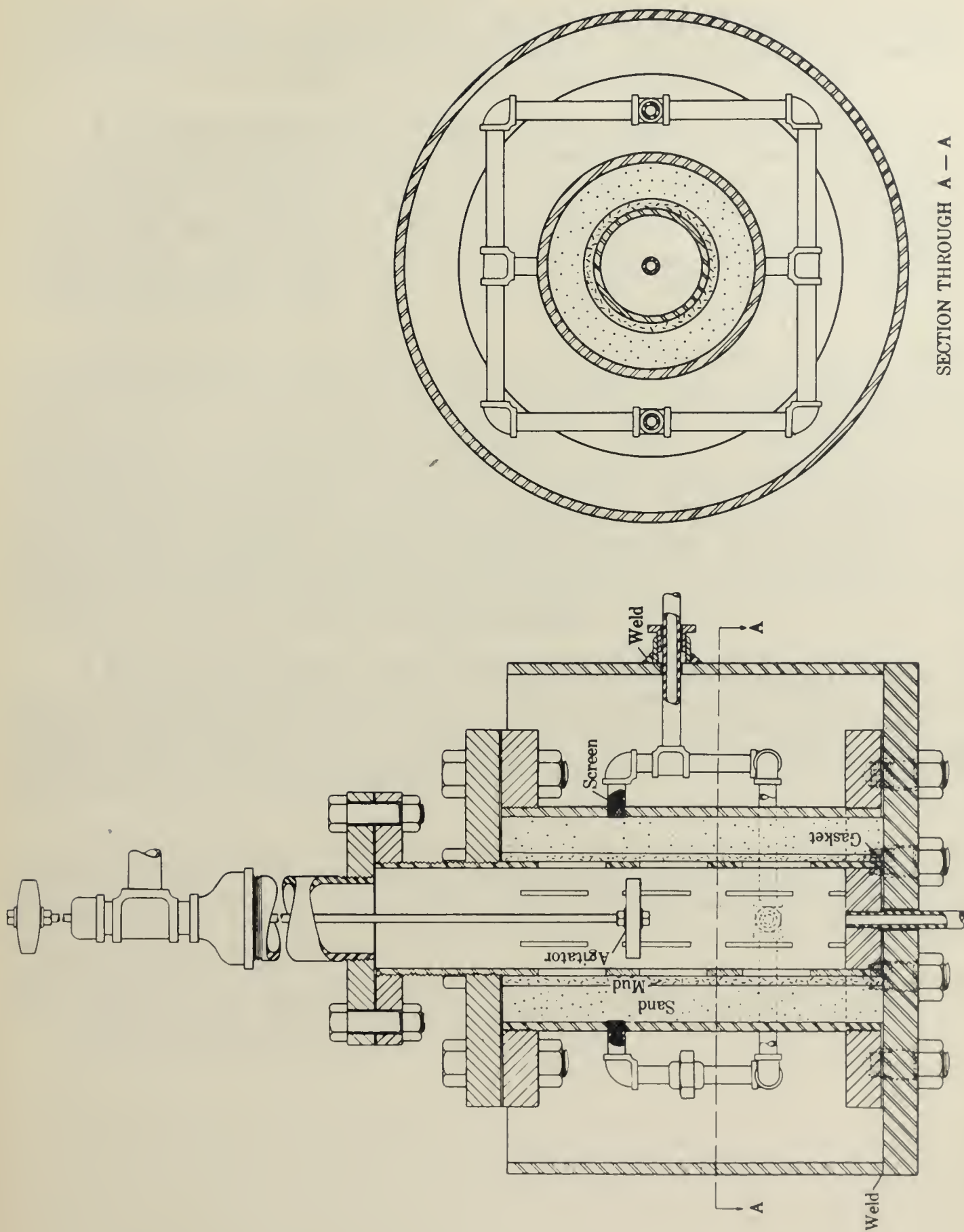


Figure 4.—Mud cake treated with 7 percent Para crude rubber: *A*, Appearance of treated cake before coming in contact with oil; *B*, cake in tube after failure; *C*, cake removed from tube; note its badly fractured condition.



SECTION THROUGH A - A

Figure 5.- Well type using sand as porous medium.

Well Type Using Sand as Porous Medium

This apparatus was a modified type of that used by Miller and Shea^{11/} in laboratory work conducted to demonstrate a chemical method for removing mud sheaths in oil wells. It is shown diagrammatically in figure 5.

A miniature well 10 inches deep was constructed from 6-inch pipe. On the upper end of the well were 6-inch companion flanges, the top one drilled and tapped for 3-inch pipe. On the lower end of the well was a 6-inch flange bolted to a piece of boiler plate 15 inches in diameter. Fifteen-inch casing approximately 12 inches long was welded to the periphery of the boiler plate. The upper end of the casing was open. The annular space between the 15-inch casing and the 6-inch pipe formed a receptacle for a constant-temperature bath.

The 6-inch pipe was drilled and tapped with four 1/4-inch holes. Two of these holes were about 3 inches from the upper end of the pipe and the other two about 3 inches from the lower end. The upper pair and lower pair were opposite, and each pair was at 90° to the other. A manifold of 1/4-inch pipe was connected to these holes, providing a common outlet to the atmosphere and making it possible to obtain a uniform differential of pressure between the well and the reservoir sand. When the manifold connections were in place, each hole was filled compactly with a strip of 20-mesh wire cloth about 1 inch in width rolled into a tight bundle. The screen was necessary to prevent sand entering the 1/4-inch manifold and clogging it when the well was under pressure.

A piece of perforated 3-inch pipe about 16 inches long was threaded at one end a sufficient length to pass through the upper 6-inch flange of the well, a distance of about 4 inches. A disk made from boiler plate was welded to the lower end of the 3-inch pipe. This disk was drilled and tapped in the center for a 1/2-inch drain pipe. The 3-inch pipe was perforated with six rows of 1/4- by 2-inch slots.

In the assembly, the upper end of the 3-inch pipe was fitted with 3-inch companion flanges, the top one of which was drilled and tapped for 2-inch pipe. A 2-inch pipe 27 inches long, constituting a reservoir, was screwed to this flange. A bell reducer was connected to the upper end of the 2-inch pipe as shown in the figure.

11/ Miller, H. C., and Shea, G. B., Chemical Method for Removing Mud Sheaths in Oil Wells: Rept. of Investigations 3249, Bureau of Mines, June 1934, p. 10.

The cap on this bell reducer formed a stuffing box, through which passed a steel rod $3/16$ inch in diameter by $1/4$ feet in length. A handle was attached to the upper end of the rod and a rubber plunger, 2 inches in diameter and $1/2$ inch in thickness, to the lower end. By moving the handle up and down the contents of the well could be agitated while under pressure.

To prepare the apparatus for an experiment, the upper 6-inch flange of the well was unbolted, and the entire perforated pipe assembly and 2-inch reservoir were removed. A mandrel made from 3- $1/2$ -inch pipe ($1/4$ inches outside diameter) was centered carefully in the well. To facilitate removal of the mandrel, one layer of oiled heavy paper was wrapped around it before it was set in place. Fine beach sand that had been thoroughly mixed with S.A.E. 30 lubricating oil was firmly packed in the annular space between the mandrel and the 6-inch pipe. The mandrel was then withdrawn by means of a block and tackle suspended from the ceiling, and the paper wrapper, which had remained in the well, was rolled carefully away from the face of the oil sand and removed. A hollow cylinder of oil sand approximately 6 inches O.D., 4 inches I.D., and 10 inches in length resulted.

The 3-inch perforated pipe assembly then was lowered carefully into the well with the aid of guides so as not to disturb the walls of the hole and landed on the gasket at the bottom. This left a $1/4$ -inch annular space between the face of the oil sand and the outer surface of the perforated pipe. The upper 6-inch flange then was bolted in place.

After the bell reducer was removed the well and reservoir were filled with mud fluid. The reducer was replaced, and then air under a pressure of 75 pounds per square inch was introduced. Agitation was begun at once and continued for an hour. The air pressure was held for 3 hours. Water from the mud fluid passed through the sand and out the $1/4$ -inch manifold. A tenacious mud sheath was formed in the $1/4$ -inch annular space surrounding the perforated pipe.

At the end of the mudding period the pressure was released and the surplus mud fluid drained from the well through the bottom outlet. The well then was washed with water to remove any mud adhering to the inner walls of the perforated pipe. The mud cake was not removed from the openings of the perforated pipe by the washing, hence the mud sheath had a thickness of $1/2$ inch at the perforations. The water was removed from the $1/4$ -inch pipe manifold by draining, and blended oil then was admitted under a pressure of 3 pounds per square inch. The assembly above the lower 3-inch flange was removed and replaced by a 3-inch blind flange. With this arrangement, inspection of the well interior was facilitated.

A 3-liter flask was placed beneath the apparatus and connected to the drain pipe by a glass S-tube. The S-tube was filled with water, so that when the blind flange was in place no air would enter the well. This prevented drying out of the mud sheath by excessive evaporation.

During the entire experiment the temperature of the water bath was maintained at 110° F. This was accomplished by a thermostatically-controlled electric heater and by circulating the water with a small pump.

The author believes that this apparatus simulates the conditions at the bottom of a low-pressure producing oil well as nearly as can be expected with laboratory equipment.

Basing judgment upon the results of the earlier experiments, crude rubber appears to be the material best suited for processing drilling mud to make it permeable to oil. For that reason rubber was the only treat material used in this type of apparatus. Two units were constructed.

Table 4 shows that an untreated mud cake was held in well no. 1 for 66 days under an oil pressure of 3 pounds per square inch. Failure was due to the developing of a small crack in the mud at a perforation, because the mud dried despite the presence of a sponge suspended in a wire cage in the well that was kept soaked with water during the entire 66 days. As mud in an oil well would not dry so rapidly, presumably a similar cake in an actual well would have held indefinitely under a pressure of 3 pounds per square inch.

In the tests recorded in the table, the mud cakes treated with 5 percent of rubber failed in an average time of 12-2/3 days. Those treated with 7-1/2 percent of rubber failed in an average time of 6-1/2 days. In all the tests recorded, failure was apparent 1 or 2 days before it took place, as mud was being pushed through the perforations by the swelling rubber.

Careful examination of the mud cake built up in all of the trials with this type of apparatus showed that the rubber was distributed quite uniformly throughout the cake. If it should seem desirable to increase the specific gravity of the rubber enough so that it will not float in water, this could be done at small additional cost by "milling" into the crude rubber when it is being processed a sufficient amount of suitable material.

Malay crude rubber of 0.91 specific gravity was milled with commercial hexachloroethane having a specific gravity of 1.8 (two parts of rubber to one part of hexachloroethane by weight). The resulting mixture had a specific gravity of 1.1. It swelled quite rapidly when immersed in gasoline; also, it was more readily soluble than the crude rubber. Hexachloroethane is a crystalline solid that is insoluble in water and readily soluble in gasoline and has a melting point of 185° C. (365° F.).

TABLE 4. - Effect of rubber on permeability of mud cake

Mojave Red Mud

Well	Treatment	Pressure, lb. per sq. in.	Lapsed time to failure (days)	Remarks
1	Untreated	3	66	Failure due to small crack at perforation.
2	5 percent rubber	[1.8 for 4 days 3 for 9 days]	13	Cake pushed out of perforations.
1	do.	3	7	Cake leaking at every perforation.
2	do.	[2 for 6 days 3 for 12 days]	18	Mud cake pushed into well.
1	7.5 percent rubber	3	9	Cake badly fractured, mud pushed through perforations.
2	do.	3	4	Partly dissolved rubber and mud cake pushed through perforations.

ESTIMATED COST OF PROCESSING MUD FLUID WITH CRUDE RUBBER

As has been stated, the laboratory work indicates that crude rubber is the best-suited material so far found for processing mud fluid in accordance with the suggested procedure of adding an oil-soluble ingredient. The following estimate is based on the use of that commodity. Five percent of rubber by weight of dry clay seems to be the minimum amount necessary to insure proper distribution throughout the mud fluid. The labor cost of introducing the rubber into the mud fluid should be small, as it could be poured into the mud sump and mixed by circulation.

Best results have been obtained when the rubber has been cut into shreds about 1/16 inch in diameter and 1/2 inch in length. As nearly as can be determined, to shred the rubber on a commercial scale will require processing by the rubber manufacturer. It has not been possible to obtain definite costs of doing this. However, rubber manufacturers who have been consulted believe that this cost will not exceed 10 cents per pound. Packing in suitable containers, freight, and cartage costs will, of course, have to be added.

Crude rubber is an extensively used commodity the price of which varies widely at times. Table 5^{12/} shows the average monthly wholesale price of crude rubber at New York from 1921 to 1935, inclusive. It will be noted that the highest price during that period was 71.9 cents per pound in 1925 and that the lowest price was reached in 1932, when it was 3.4 cents per pound. The price of crude rubber at the present time (April 1937) at New York is around 25 cents per pound.

12/ Compiled from data given in 1936 supplement, U. S. Bureau of Foreign and Domestic Commerce, Survey of Current Business, p. 133.

TABLE 5. - Wholesale price of crude rubber
(smoked sheet) at New York

Year	Monthly average price, cents per pound
1921.....	16.4
1922.....	17.4
1923.....	29.6
1924.....	26.1
1925.....	71.9
1926.....	48.5
1927.....	37.6
1928.....	22.3
1929.....	20.5
1930.....	11.9
1931.....	06.1
1932.....	03.4
1933.....	06.0
1934.....	12.9
1935.....	12.4

For the purpose of this estimate it will be assumed that the rubber in the desired form can be laid down at the well for 50 cents per pound.

Table 6 shows the average cost of drilling oil wells in different fields of the United States during 1931. From this table it will be seen that to drill an average well 3,300 feet deep cost in that year about \$27,400.00.

TABLE 6. - Cost of drilling oil wells in the
United States during 1931^{1/}

Field or District	Depth of well, feet	Total cost of drilling (not including pumping equipment)
Central Ohio and West Virginia.....	3,000	\$14,976.50
East Texas.....	3,650	23,746.13
North and West Texas.....	3,500	37,782.51
Wyoming.....	3,000	32,930.70
Average.....	3,287	\$27,356.71

^{1/} Compiled from data given by Walter H. Jeffery in Deep Well Drilling, 3rd Edition (rev.), Gulf Publishing Co., 1931, p. 689.

It has been found by experiment that 5 pounds of Mojave (Calif.) clay mixed with 1 gallon of water makes a drilling mud weighing 81 pounds per cubic foot. One gallon of water weighs 8.33 pounds. The total weight of the clay and the water would be 13.33 pounds. One cubic foot of 81-pound drilling mud, therefore, contains:

$$81 \times \frac{5}{13.33} = 30.4 \text{ pounds of clay per cubic foot of mud fluid.}$$

Assuming the top of the oil sand to be at a depth of 3,300 feet and the hole to be finished with 8-inch casing, the cost of processing the mud fluid necessary to "bring in" the well may be estimated approximately as follows:

The cubical content of the well will be

$$3,300 \times 0.3491 \text{ (cubic foot per foot of length of 8-inch casing)} = 1,152.0 \text{ cubic feet.}$$

This also will be the approximate amount of mud fluid to be processed to "bring in" the well. The quantity of clay required would be $30.4 \times 1,152 = 35,000$ pounds, to make up the mud fluid in the hole. For the purpose of easy calculation, the quantity of rubber required will be computed as 5 percent of the clay, or $35,000 \times 0.05 = 1,750$ pounds. Its cost would be $1,750 \times \$0.50 = \875.00 , and would represent $\frac{875.00}{27,400} \times 100 = 3.19$ percent of drilling cost.

Allowing 20 percent for adequate reserve of processed mud while drilling would increase the percentage of drilling cost to 3.83 percent.

A. H. Bell, production engineer, Continental Oil Co., has advised the author that the average cost of drilling a 6,000-foot well finished with 8-inch casing in the Los Angeles (Calif.) basin fields is about \$100,000, and such a well requires approximately 50 tons of clay for mud fluid to drill its entire depth.

On this basis, the extra cost incident to the use of a rubber-treated mud during the entire drilling of the hole would be \$2,500 or 2.5 percent of the total. If a rubber-treated mud were used only while drilling through the productive horizon at the bottom of the hole, the cost of processing the mud fluid would be \$1,900, or 1.91 percent of total drilling cost as calculated by the method of the previous example.

It should be possible to reclaim rubber, even though it has been circulated, if it has not become contaminated by oil. This may result in a considerable saving, especially where several wells are to be drilled.

CONCLUSIONS

Although it is too much to hope that laboratory experiments will ever accurately reproduce underground oil-well conditions, there seems to be no doubt that drilling mud will solidify when in quiescent contact with crude oil. The time required for such solidification to take place and the degree of hardening attained will depend greatly on the amount of moisture in the mud cake and the amount and type of the oil that is contacted.

The author believes that in many producing fields and in "wildcat" wells low-pressure oil- and gas-bearing formations have been drilled through without being detected. Because the mud sheath on the wall of the well has solidified, the formations are now as securely sealed as though they had never been penetrated. He also believes that many prolific oil sands are yielding only a small part of their oil as they have become clogged with a solidified mud cake on the face of the sand.

In the past some operators tended to have their drillers "make hole" too rapidly and neglect testing formations that may have possibilities. More careful testing of each formation as the drilling proceeds and the use of a mud fluid properly processed to permit more of the oil and gas encountered to flow freely into the well seems desirable in both proved and "wildcat" territory.

The use of processed mud fluid in "bringing in" a well should insure a maximum production of oil because the treat material will shatter the mud sheath if it tends to harden. If a proper amount of treat material is used the mud sheath should be permeable to oil, perhaps as permeable as the oil-bearing formation.

The cost of processing the mud fluid should not be excessive. If several oil- and gas-bearing strata are encountered during the course of drilling, it should be profitable to use a treated mud throughout. When the deeper zones have been exhausted or found nonproductive it will be possible to seal them by "plugging back" and obtain the production from the upper low-pressure zones, because the mud sheath opposite the oil-bearing strata will be permeable.

It should be remembered that this report is based on laboratory experimental work. The practicability of the proposed method of minimizing the sealing effect of mud sheaths in an oil well has not yet been tested in the field; neither has the practicability of circulating a mud treated with such a material as short strips of crude rubber been demonstrated. Operators with whom field application of the process has been discussed are of the opinion that a definite technique covering its application will have to be established. They do not, however, foresee insurmountable difficulties.

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REPORT OF INVESTIGATIONS

DEPARTMENT OF THE INTERIOR - BUREAU OF MINES

STUDIES OF ROOF MOVEMENT IN COAL MINES

1. MONTGOMERY 10 MINE OF THE PITTSBURGH COAL CO.^{1/}

By H. P. Greenwald,^{2/} E. R. Maize,^{3/}
Irving Hartmann,^{4/} and G. S. Rice^{5/}

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- ^{1/} The Bureau of Mines will welcome reprinting of this paper, provided the following footnote acknowledgment is used: "Reprinted from U. S. Bureau of Mines Report of Investigations 3355."
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FOREWORD

By George S. Rice

Accidents from falls of roof in coal mining are and always have been the major cause of fatalities and injuries in the coal mines in every coal-mining country, accounting for 40 to 50 percent of all the deaths and injuries. Studies of roof movement were an important part of mining investigations begun under the U. S. Geological Survey in 1908 and transferred to the Bureau of Mines on its establishment in 1910. After general investigations of roof conditions and existing mining methods in the coal industry, the first method of attack was an educational campaign advocating wider use of the best methods of roof support already known by mining men but not always practiced. Papers of an educational nature on timbering and other means of support were issued. The formulation of recommendations for better timbering and the use of other roof supports has been a continuing Bureau problem.

In 1909-11 the Bureau acted in an advisory capacity to a commission of the city of Scranton, Pa., appointed to determine remedies for the serious caving within the city limits that had occurred from time to time, endangering life as well as property. The report of the engineers engaged by the commission, Eli T. Connor and William Griffith, was published by the U. S. Bureau of Mines with a foreword by Dr. Joseph A. Holmes, the first director of the Bureau, and with a chapter by N. H. Darton on the availability of backfilling material^{6/}.

Following this investigation, the State of Pennsylvania established a commission to make a general investigation in the anthracite region to determine ways of supporting mine workings, Charles Enzian, an engineer of the Bureau, acting as secretary. As a part of the program, research was undertaken by the Bureau on the bearing strength and compressibility of the usual kind of mine supports. Samples of various supports, including full-sized props and timbers, were sent from the anthracite mines to the experiment station of the Bureau at Pittsburgh. A report was submitted by the Bureau to the Pennsylvania State Commission in 1913; this report included some test data on the bearing strengths of small and large blocks of bituminous coal.

However, the report was not published by the State, and, as there was much demand for the information contained in it, an abstract was prepared by the writer and published by the Bureau of Mines^{7/}. In an introductory review, the writer expressed the belief (developed since the testing was done in 1912) that laboratory determinations of the bearing strength of coal were not indicative of the strength of pillars in a mine, because of the absence of lateral restraint. Because of this, he proposed that tests of the bearing strength of coal be made in the mine on the coal in place,

^{6/} See appendix, reference 22.

^{7/} See appendix, reference 62.

that is, unmined. On this basis, the first data were secured incident to making tests of the strength of concrete stoppings designed to resist explosions in coal mines. The data are given in Bulletin 345 and Technical Paper 527^{8/}.

Interspersed in these tests of the lateral compressibility of coal were some preliminary tests of a similar character made in a vertical direction. The procedure was to excavate a space over the Pittsburgh bed in the experimental mine and place in it vertically a hydraulic jack buttressed against the roof and pressing on a square iron plate of definite size, which bore on the coal bed. Depression of the corners of the plate under successively higher loads was measured, together with movement of the free face of the coal bed exposed at the side of the passageway. This testing of the vertical compressibility of the coal had to be suspended for lack of funds but will be resumed when opportunity offers.

Important studies of ground movement and methods of support of workings were carried on in cooperation with the Illinois Geological Survey and the Engineering Experiment Station of the University of Illinois, 1913 to 1916. Among the more important contributors were Dr. L. E. Young, then connected with the mining department of the University of Illinois, the late H. H. Stock, and Professor C. M. Young. Reports of these studies were published by the Engineering Experiment Station of the University of Illinois^{9/}. The Bureau of Mines staff, besides acting in an advisory way in the above studies, started with members of its local Illinois staff a special long-continuing investigation of ground movement and surface subsidence; this was begun in 1916 with establishment of monuments in four districts having different geographical conditions and methods of mining. The study in the case of longwall mining also concerned questions of gob walls and backfilling at the longwall face. This work, extending over a 5-year period, was carried on successively by Messrs. H. I. Smith, W. B. Plank, C. A. Herbert, and J. J. Rutledge. The two latter presented the final report published in full as Bureau of Mines Bulletin 238^{10/}.

Through a special appropriation for an investigation of falls of roof and preventive methods, studies of methods of coal-mine timbering

^{8/} Rice, G. S., Greenwald, H. P., Howarth, H. C., and Avins, S., Concrete Stoppings in Coal Mines for Resisting Explosions. Detailed Tests of Typical Stoppings and Strength of Coal as a Buttress: Bull. 345, Bureau of Mines, 1931, 63 pp.

Greenwald, H. P., Avins, S., and Rice, G. S., Compressibility and Bearing Strength of Coal in Place. Tests of Lateral Compression of the Pittsburgh Coal Bed: Tech. Paper 527, Bureau of Mines, 1933, 12 pp.

^{9/} Young, L. E., Surface Subsidence in Illinois Resulting from Coal Mining: Bull. 17, Cooperative Coal-Mining Series, Illinois State Geol. Survey, 1916, 122 pp. Also appendix, reference 47.

^{10/} See appendix, reference 23.

were conducted in many of the coal fields of the country under the guidance of J. W. Paul, and a series of Bureau papers was published^{11/}.

The writer of this foreword was commissioned, on several trips to Europe for specific investigations, to look into the question of roof supports, and has presented various papers on the subject. A general report on European coal-mining methods, including studies of mine supports and of backfilling of mining excavations by hydraulic and pneumatic agencies is in course of preparation for publication by the Bureau of Mines.

About 1930, the Safety in Mines Research Board of Great Britain, in cooperation with committees of the British Institution of Mining Engineers, began studies of ground movement at working places and employed a method of measuring the convergence of roof and floor following the advance of the working face. Measurements of this character were supported by leveling to determine how much of the convergence was due to the descent of the roof and how much to the rise of the floor. The Board also made large-scale laboratory tests somewhat similar to those made by the Bureau in 1912, on the bearing strength of timber cogs and cribs. A large number of papers describing all this work have been issued^{12/}.

In certain instances the Board's investigations included attempts to measure the load imposed by the roof on supports in working places. Recording and indicating dynamometer props were used for this purpose. In the cases reported, the results were not conclusive in the opinion of the writer of this foreword, as the dynamometer prop was surrounded by a group of ordinary props of different mechanical characteristics. To obtain correct measures of the loading, it seems necessary that all the props in a considerable area, the extent depending on the conditions, should be dynamometer props.

Meanwhile, for determination of vertical movement of roof and floor, convergence recorders were recommended for trial in certain mining operations in the United States. Such studies are under way in the United States Potash Co.'s mine at Carlsbad, N. Mex., and in an eastern Kentucky coal mine in connection with a study for the prevention of disastrous bumps^{13/}.

^{11/} Preliminary publication was in the form of reports of investigations. The work was summarized later by a series of technical papers under the authorship of J. W. Paul, J. N. Geyer, L. N. Plein, and J. G. Calverley. See Bureau of Mines Technical Papers 520, 534, 541, 547, 550, and 563 issued between 1932 and 1935.

^{12/} See appendix, references 13-16, 18-21, 24, 25, 27, 29-35, 38-41, and 46.

^{13/} Bryson, J. F., Methods of Eliminating Coal Bumps or Minimizing their Effects: Trans., Am. Inst. Min. and Met. Eng., vol. 119, 1936, pp. 40-57.

Rice, G. S., Bumps in Coal Mines of the Cumberland Field, Kentucky and Virginia: Causes and Remedy. Report of Investigations 3267, Bureau of Mines, 1934, 36 pp.

Records were also obtained in a mine in Nova Scotia which had experienced bumps when the depth of workings in a dipping bed had reached over 2,000 feet, and to prevent or lessen their occurrence the writer had recommended a longwall retreating system^{14/}.

Dr. H. Landsberg, of the State College of Pennsylvania, also has carried on tests in a coal mine in central Pennsylvania, using a convergence recorder of his own design but similar to the British convergence recorder^{15/}.

The first extensive testing of convergence of roof and floor carried on by the Bureau of Mines staff, and its relation to the working face and the subsidence caused on the surface, was undertaken with the cooperation of the Pittsburgh Coal Co. at its Montour 10 mine in the Pittsburgh coal bed, as described in the following report. The field work was carried on by Bureau engineers with the assistance of officials of the Pittsburgh Coal Co. as detailed in the body of the report. This study deals with a particular mining method in a highly mechanized working and under particular conditions in the Pittsburgh coal bed. It is hoped that other investigations of a similar nature may be undertaken under other mining conditions, with the ultimate objective of obtaining indications which will permit better roof control either by a change in mining method or by the use of special supports appropriate to the mining method and natural conditions.

GEORGE S. RICE
Chief Mining Engineer

^{14/} See appendix, reference 28.

^{15/} See appendix, reference 26.

INTRODUCTION

In many coal-mining countries, during the past decade, there have been made extended studies and accurate measurement of movement of roof strata, tests of the physical properties of specimens cut therefrom, and tests of the strength of and load on artificial supports. Naturally, such studies have been more extensive in European countries, where mining conditions are less favorable than in the United States. As a part of its work in the conservation of coal resources and prevention of accidents in mines, the Bureau of Mines began systematic measurement of the movement of roof strata in coal mines in the fall of 1935.

It was decided that, in the beginning, work on a rapidly retreating rib line in a mechanized mine would give the greatest amount of information in a reasonable time. To this end a cooperative agreement was made with the Pittsburgh Coal Co., through Dr. L. E. Young, vice president in charge of operations, for conduct of work in that company's Montour 10 mine operating in the Pittsburgh coal bed. This paper gives an account of the methods used in the investigation, excerpts from the data obtained, and such generalizations as appear to be justified.

There was prepared concurrently a digest of the technical literature describing similar work in different countries, and from this has been abstracted the short selected bibliography appearing in an appendix. This bibliography covers about 10 percent of the references reviewed but is believed to include the more valuable papers that have been published in English and can be obtained readily by American mining men. References to this bibliography are by the numbers assigned to the papers therein.

ORGANIZATION AND ACKNOWLEDGMENTS

Development of plans and selection of the site of the work were in the hands of George S. Rice, chief mining engineer, and H. P. Greenwald, supervising engineer of the Experimental Coal Mine Section; the latter was also in general charge of the work during its progress. The entire staff of the section was engaged in the work at different times; these men are H. C. Howarth, coal mine superintendent; E. R. Maize, assistant mining engineer; Irving Hartmann, assistant physicist; Ray Wilson, mine electrician; and D. C. Thomas and John Nagy, junior physical science aids. Maize was in direct charge of the work underground and spent his time with it continuously, assisted by other members of the staff as needed. He and Wilson assisted Greenwald in compression tests of posts used in the mine. Hartmann made a search of literature dealing with the subject, on which is based the appended bibliography. Assisted by Maize, he also conducted the tests of rock specimens obtained. Petrographic examination of these rocks was made by G. C. Sprunk of the Coal Constitution Section. Such analyses as were needed were made by the Coal and Miscellaneous Analysis Sections, H. M. Cooper and W. A. Selvig, supervising chemists. Most of the special equipment required was built in the Bureau's shops.

Officials of the Pittsburgh Coal Co. were interested in the work at all times and success was possible a number of times only through their aid. Dr. Young kept in touch with the work as it progressed and directed that all needed assistance be given. The supervisory staff at Montour 10 mine with whom the investigators worked were W. H. Barker, superintendent; H. C. Rose, assistant superintendent; James Humes, mine foreman; William Feick, Andrew Boyd, and Mac Riggs, assistant foremen; and Mike Gary, Alex McClymont, and James Campbell, section foremen. All these men followed the work with interest and rendered assistance whenever required. At one time, when press of work was such that the Bureau men could not cope with it, Dr. Young employed Anton Bogarty as an assistant to Mr. Maize for a period of 6 weeks. In the latter part of the work it was agreed that measurements of surface subsidence should be made parallel with the underground studies. The surface points were established and weekly leveling was conducted by a corps of the company's engineers under the direction of James Kerr.

The authors are much indebted to Professors F. M. McCullough and J. M. Daniels of the Department of Civil Engineering, Carnegie Institute of Technology, for use of and assistance with the equipment in the department's materials testing laboratory, which was used in tests of rock and timber specimens and in calibration of dynamometer props.

SITE OF THE WORK

Montour 10 mine, working the Pittsburgh coal bed, is located at Library in the southern end of Allegheny County, Pa., close to the Washington County line. At this point the coal has been eroded from the valley of Piney Fork, but the bed dips slightly and the ground rises in a general westerly direction so that a short distance up the valley the bed is continuous. The following paragraphs are descriptive of conditions as they were at the time of the present investigation.

In the newer workings quadruple face entries are turned at such intervals and pairs of butt entries connect them in such manner that the coal is divided into blocks approximately 300 by 2,000 feet in size, exclusive of the amount in entry chain and barrier pillars. The 2,000-foot dimension is parallel to the face cleat of the coal and in a general east to west direction. All face and butt entries are numbered, but groups or pairs are commonly known by the numbers of those on which haulage is done. Thus, the present work was done in the area served by 15, 17, 19, and 21 butts between 29 and 33 face entries.

Each large block of coal is opened by rooms 15 feet wide driven north from the odd-numbered butt entries toward the even-numbered entry of the preceding pair. These rooms are advanced in sequence from east to west and retreat on the pillar to the east of each room is started as soon as a room is completed. Rooms from two butt entries are worked together to give a continuous and stepped diagonal rib line approximately 950 feet long. At the start of retreat on a pillar a crosscut is driven through to the preceding room behind a narrow fender. This fender serves primarily as a safe support

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INCHES

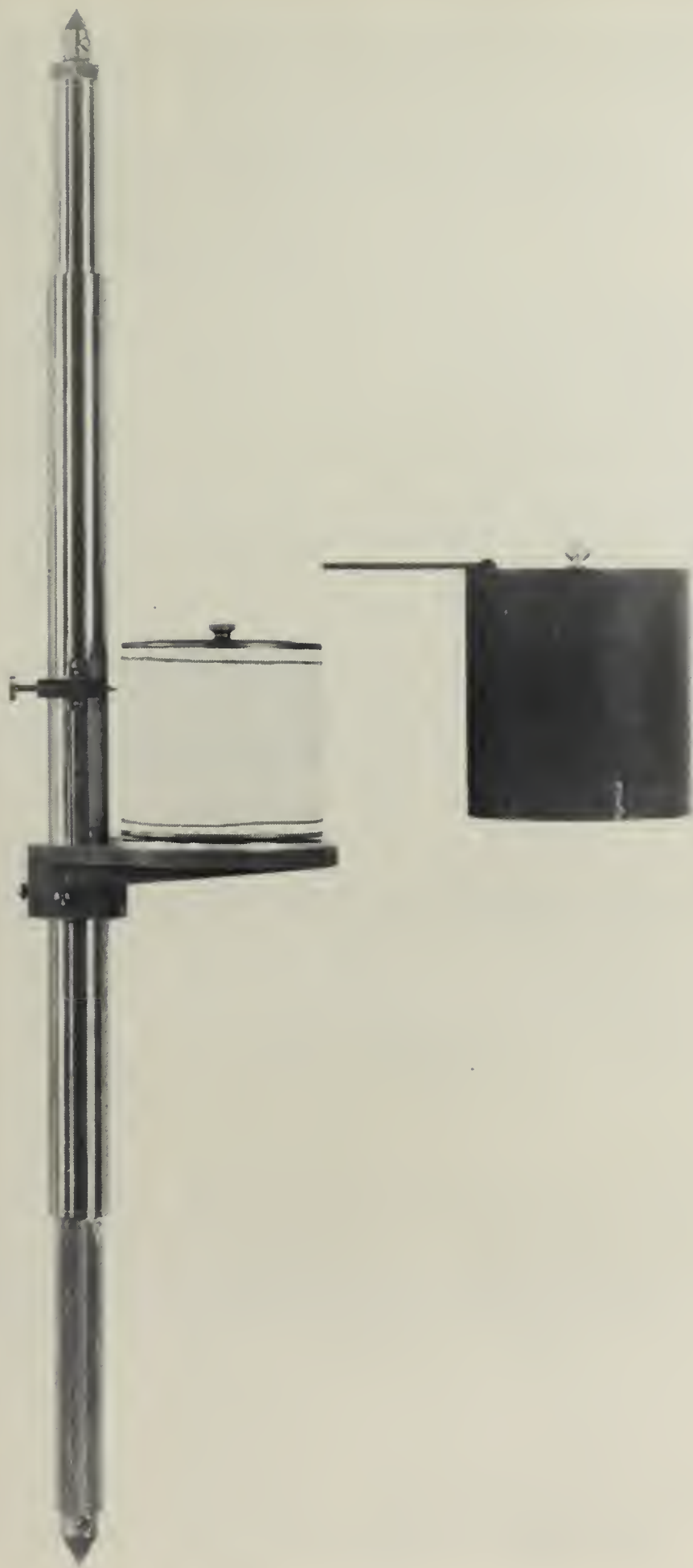


Figure 1.— Bureau of Mines type convergence recorder.

for the roof strata that have broken and fallen in the space beyond; secondarily, it keeps the gob out of the working space. When this cross-cut is finished slab cuts are taken on the new face thus formed until this procedure is stopped either by the weight on the coal or by the condition of the roof. A new crosscut is then started behind another fender. Coal is moved from the rooms and along the butt entries mechanically.

GENERAL SCOPE OF THE WORK

In order to obtain as extensive information as possible, the measurements of strata movement underground were supplemented by mechanical tests of such rock specimens as could be obtained, mechanical tests of posts used, a study of the stratigraphy of the ground, and a careful record of the progress of mining. Work underground was concentrated on measurement of vertical movement of the strata, as it was evident that lateral movement was small except in areas on the goaf side of the rib line that could not be reached regularly and safely because of impending falls. Because of the firmness of the floor, it was judged that practically all movement would be in the roof and coal and relative or convergence measurements would be satisfactory. The immediate floor is a thin fireclay (rarely over 4 inches thick), below which is a stratum of hard limestone 12 to 18 inches thick.

Attempts were made also to measure the load on timbers through dynamometer props used in place of the regular posts.

Instruments Used

To measure relative movement, holes were drilled in roof and floor at the selected points, the one directly above the other. Those in the floor were carried well into the limestone stratum. Those in the roof were 18 inches deep in order to get into as solid strata as possible. This was particularly necessary in rooms in which the draw slate is not taken down but is carried on posts and crossbars. Originally, iron pins were cemented into the holes, but later it was found advantageous to substitute long machine bolts with expansion shields. The bolt then moved with the stratum in which the end was fixed.

Graphic records of the relative vertical movements of roof and floor were obtained with the convergence recorder shown in figure 1. This is similar in general design to the original Safety in Mines Research Board instrument (reference 14) and consists of two telescoping brass tubes held, when free, in extended position by a strong spring. Aluminum extension pieces permit spanning any height. Clamped to the outer tube is a bracket carrying a drum-type clock making one revolution per week. Fastened to the inner tube is a carrier holding a pencil or stylus that draws the record on a chart attached to the clock drum.

Observations were made also with a telescoping rod on which was mounted a scale, the investigator carrying the rod with him on his rounds. Where movement at one point was to be read at frequent intervals, a third instrument called a convergence indicator was used. In this instrument the bracket

and clock of the recorder were omitted and a scale was fastened to the outer tube and a pointer to the inner tube.

The first dynamometer prop constructed depended for its operation on the elastic compression of steel. It was found that a measuring gage combining sufficient sensitivity with rapidity of manipulation and ruggedness could not be produced readily. Furthermore, the compression characteristics of steel are so radically different from those of the wood posts used that information given by this prop could not be taken to represent the load carried by a regular wooden post. The wood should be retained insofar as possible, and it was decided that this could be done best by placing a small hydraulic jack on the floor with the post resting on it. If the jack is sealed so that no liquid can escape, the pressure in the jack is a direct measure of the load thereon. Experience has shown that this system can be made to work, but the best method of measuring the liquid pressure remains to be determined. It is essential that there be a minimum of liquid movement in the jack. A Bourdon-tube-type pressure gage is not sufficiently rugged, and it is necessary to make the best possible compromise between ruggedness, minimum of movement, ease of reading, and sensitivity.

NATURE AND PHYSICAL PROPERTIES OF THE STRATA

Columnar Section

Figure 2 is a columnar section of the strata from the Pittsburgh coal bed to the surface plotted from the log of a diamond-drill hole furnished from the company's records by Dr. Young. The hole was approximately one-half mile from the site of the present work and the cover was somewhat thinner at that point. The column is divided into 12 sections, thinner and less important beds being grouped together where adjacent.

Immediate Roof

The immediate roof (sec. 2 of fig. 2) consists of nine thin layers of coal and slate; these include the draw slate immediately over the Pittsburgh coal. The tendency of the draw slate to fall as soon as the coal is removed is common knowledge. In the present case it was taken down in entries but not in room-and-pillar workings, being carried on posts and crossbars instead. It is also common knowledge that the other eight of the nine thin beds forming the immediate roof strata usually require little or no timbering in narrow workings for long periods after they are exposed. Individually, they are much too thin to support their own weight over a wide place and are responsible for local falls of roof after the draw slate has been taken down. The thickness of these nine layers is variable and in particular the thickness of the draw slate varies widely. In the area studied the draw slate was thin and the total thickness of the nine layers was one to two feet less than in the section shown in figure 2.

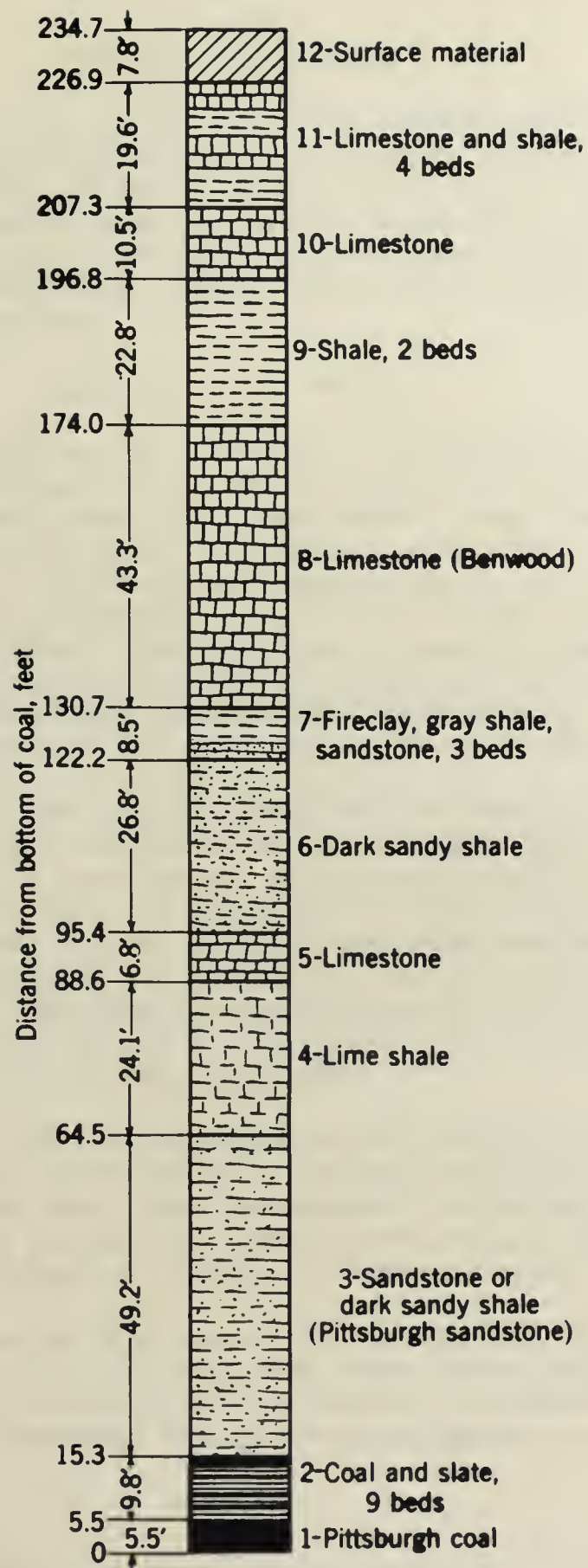


Figure 2.—Section of strata above Pittsburgh coal bed near site of investigation.

Pittsburgh Sandstone

Section 3 of the column in figure 2 is a massive sandstone identified as the Pittsburgh sandstone on the basis of a U. S. Geological Survey publication. The generalized columnar section given in U. S. G. S. folio 177^{16/} shows the Pittsburgh sandstone member immediately above the Pittsburgh coal and states that it is "shale and sandstone, in part crossbedded." Montour 10 mine is in the southeastern corner of the Carnegie quadrangle to which this reference applies.

Examination in the Mine

The Pittsburgh sandstone was examined at a number of places where it could be reached by climbing falls. This examination was made by Maize accompanied at different times by Rose and Humes of the mine staff. Two distinct kinds of rock were found in different localities. The first of these was fine-textured, uniformly dark gray to black, and appeared more like a very hard shale than a sandstone. This was the nature of the rock immediately over the area studied. The second was evidently a light-colored sandstone with thin and irregular interbedded streaks of carbonaceous material which, individually, were small in area and discontinuous. Careful search was made for slips or cleats, but only one was found in all the areas studied.

These observations were possible only because many of the major falls along rib lines were not in contact with the unbroken rock. It was possible to climb 15 to 25 feet in some places, and a beam of light could be thrown along the crack much farther. It was observed that the angle of break varied from 30° to 60° from the vertical with the dark gray rock and 10° to 20° with the sandstone, inclining over the goaf in all cases. All breaks observed were rough and jagged. Rib lines in one portion of the mine are at right angles to those elsewhere, and it was reported that no difference in behavior of the roof had been noted in the two places.

Near the close of the investigation it was possible to inspect the lower portion of this rock at a point where a haulage road was being graded across a depression in the coal bed. The light-colored sandstone was present, and the lower 15 feet of it was hard and massive with only widely scattered local and minor evidence of slips and pronounced bedding planes.

The general conclusion from these observations is that both types of rock may be expected to act as a unit over their entire thickness insofar as bending is concerned, at least up to the point of incipient failure. Large specimens of each were obtained during the investigation for examination in the laboratory.

16/ Shaw, E. W., and Munn, M. J., Geologic Atlas of the United States, Burgettstown-Carnegie Folio, Pennsylvania: Folio 177, U. S. Geological Survey, 1911, p. 17.

Petrographic Examination

Portions of the specimens obtained were examined petrographically. The light-colored rock was termed sandstone and the dark-colored rock shale, and the following was reported concerning them:

The sandstone consists of quartz and mica grains with a clay binder. The average size of quartz and mica grains is 122 microns (0.005 inch) and 310 microns (0.012 inch), respectively. The shale is exceedingly fine grained, containing approximately 40 percent of tiny quartz grains embedded in a soft shaly material with fair amount of mica, carbonaceous material, and pyrite. Size of quartz grains is 6 to 7 microns (0.00023 to 0.00027 inch).

Chemical Analysis

Samples of the two rocks were analyzed for certain constituents, which were reported as follows:

<u>Constituent, percent</u>	<u>Sandstone</u>	<u>Shale</u>
Combined water	3.1	2.8
Organic hydrogen	.2	.2
Organic carbon	.8	1.1
Carbon dioxide	4.6	22.0

These analyses are much alike except for the carbon dioxide content. The bond in the sandstone is siliceous or clayey, but the shale has a carbonate bond. Evidently it is not correct to call this rock a shale, but the name is convenient to distinguish it from the true sandstone. There is only slight difference in the quantity of organic material in the two rocks, but in the shale this material is very finely divided and uniformly distributed, which accounts for the difference in color. In sum, the major differences in the two rocks lie in the nature of the bond and in size of the particles bonded.

Strength Tests

Three rectangular specimens were cut for beam tests - two from the sandstone and one from the shale. The dimensions of these specimens were:

	<u>Sandstone</u>		<u>Shale</u>
	1	2	
Length, inches	14.06	20.06	18.06
Width, inches	5.03	5.98	5.98
Depth, inches	2.02	2.02	1.94



Figure 3.— Rock beams and cubes after test to failure.

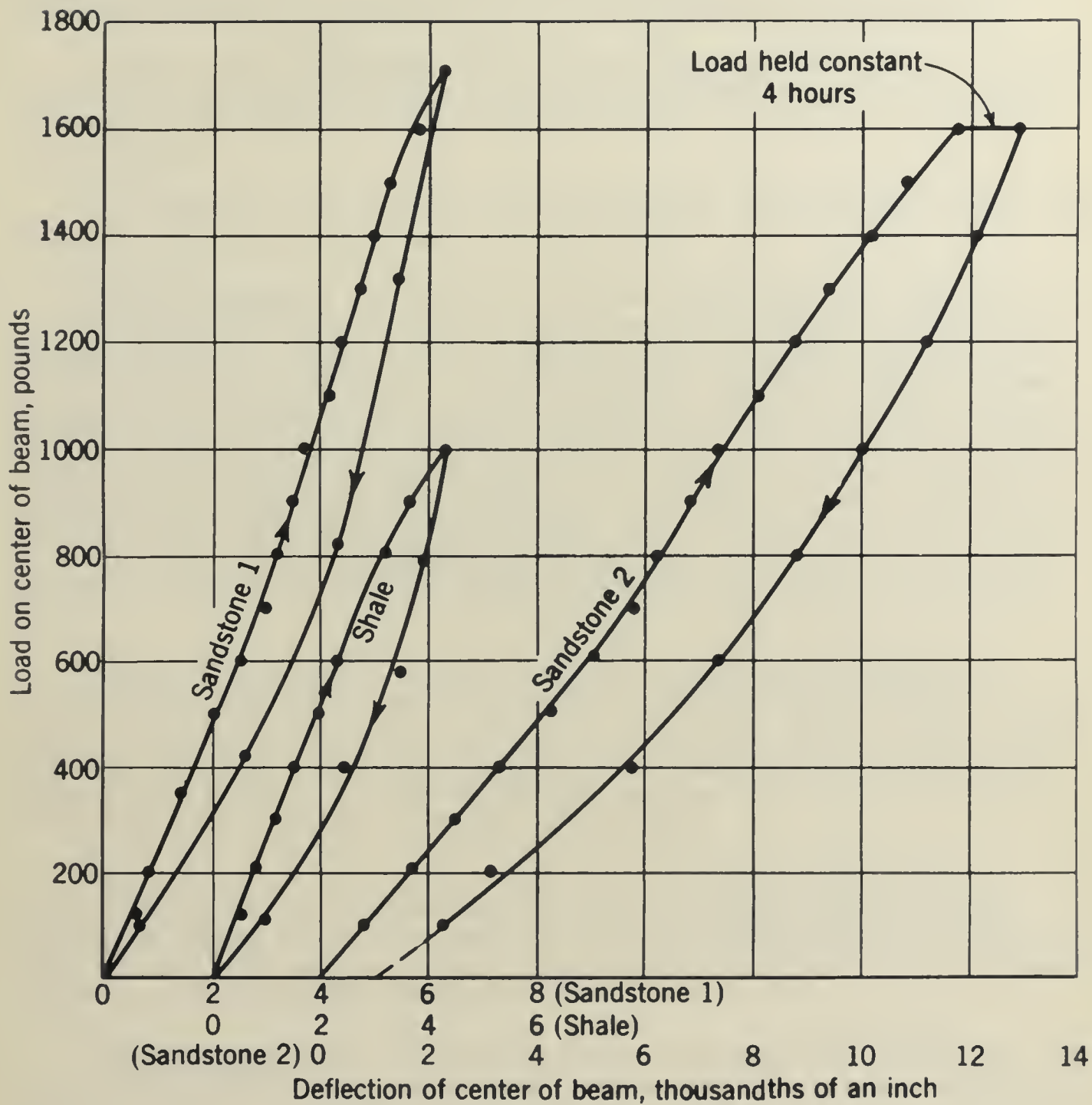


Figure 4.— Typical data from tests of rock beams.

These were tested as simply supported beams with concentrated load at the center. The free span was 12 inches for the first sandstone and the shale beam and 18 inches for the second sandstone beam. Two sandstone cubes, closely 2 inches on a side, were prepared also and crushed, the first with load perpendicular and the second with load parallel to the bedding.

Figure 3 is a photograph of all these specimens after test to failure. The sandstone beams are at the left, the shale beam is next to the right, followed by the cubes broken by compression perpendicular and parallel to the bedding.

The density of the two rocks was determined; the sandstone weighed 167 pounds and the shale 190 pounds per cubic foot.

Tests of Beams. - The tests of the three beams were too extensive to be given here in detail. Both the sandstone and the shale are brittle elastic materials. When a load is first applied and then removed, beams receive some permanent set, but additional rapid application and release of load do not increase this set to an important extent. Figure 4 shows typical curves obtained by plotting deflection of the center of the beams against total load on them. Recovery of deflection lags when the load is released and a hysteresis loop is formed. The maximum load on sandstone beam No. 2 was held constant for 4 hours in the cycle plotted, and deflection continued at a decreasing rate during that time. When the load was removed a large share of this increased deflection remained as permanent set.

Moduli of elasticity computed from the three curves range from 2.3×10^6 (2,300,000) to 3.6×10^6 pounds per square inch. Probably it is best to take 3.0×10^6 as a general value applicable to both rocks. This value is one-tenth that of steel, which means that prior to rupture the rocks may be expected to deflect 10 times as much as steel would deflect under similar conditions of support and loading.

The three beams were tested to failure finally and the moduli of rupture computed. The sandstone beams gave an average value of 2,210 pounds per square inch, as compared with only 990 pounds for the shale beam. These figures give an idea of the relative strength of the two rocks when subject to cross-bending stresses.

Compression Tests. - The sandstone cube compressed perpendicular to the bedding failed at 19,400 pounds per square inch. The cube compressed parallel to the bedding failed at 14,000 pounds. As figure 3 shows, compression perpendicular to the bedding caused failure in the double pyramid form frequently obtained with brittle materials, but compression parallel to the bedding caused the cube to split along the bedding planes. The rocks are weak in shear parallel to the bedding, and this is particularly true of the shale.

Rocks Above Pittsburgh Sandstone

Figure 2 shows 15 different beds between the Pittsburgh sandstone and the surface. Except for the unconsolidated surface material, these were inaccessible during the investigation and the part they play in roof movement can only be inferred from their thickness, the known properties of similar rocks, and the load they support. An analysis along these lines indicates that section 8 of the column (fig. 2) is the only bed of importance above the Pittsburgh sandstone insofar as strength is concerned.

From the general columnar section referred to in footnote 16, it is likely that the bed of limestone marked section 8 on figure 2 is the Benwood, which is reported as composed of a number of thin white, gray, and blue limestones separated by thin beds of calcareous shale. From a strength standpoint this possible division into a series of thinner beds or the presence of continuous vertical joint planes is important. If these horizontal or vertical planes of weakness are present, the entire mass will certainly subside with the lower beds as soon as the Pittsburgh sandstone collapses. On the other hand, if the limestone is massive it can span a considerable area because of its thickness and the light weight of the overburden. The only evidence comes from records of surface subsidence (described later) and favors the idea that this limestone bed subsides promptly as soon as the Pittsburgh sandstone breaks.

Summary

The foregoing data on the strata over the Pittsburgh coal near the site of the present investigation show that one should expect the Pittsburgh sandstone to be the principal factor governing major falls along rib lines. Because of its thickness and massive nature, it is very strong and can span large areas without breaking even when the total cover over the coal reaches 300 feet. Its elastic properties will permit it to bend considerably before the breaking point is reached, as should the fine-grained black shale that replaces it over some parts of Montour 10 mine. This shale is not as strong as the true sandstone and should fail on shorter spans. Once these main rocks break, it is to be expected that subsidence will proceed to the surface rapidly.

Compressive Strength of Pittsburgh Coal

Tests of the compressive strength of Pittsburgh coal are few in number, and there is some question as to how the data should be interpreted. Rice and Enzian (reference 62) showed that the strength of cubes decreased rapidly as the size of the cube increased. Cubes 2-1/2 to 4 inches had an average crushing strength of approximately 2,500 pounds per square inch, but a 30-inch cube failed at 817 pounds and a 54-inch cube at 306 pounds. The larger cubes failed by disintegration along the cleats across which the coal has negligible tensile strength. The authors have access to the original reports of these tests, and figure 5 shows the relation of compression to load obtained with the 30-inch cube; in this figure the load is given in tons per square foot.

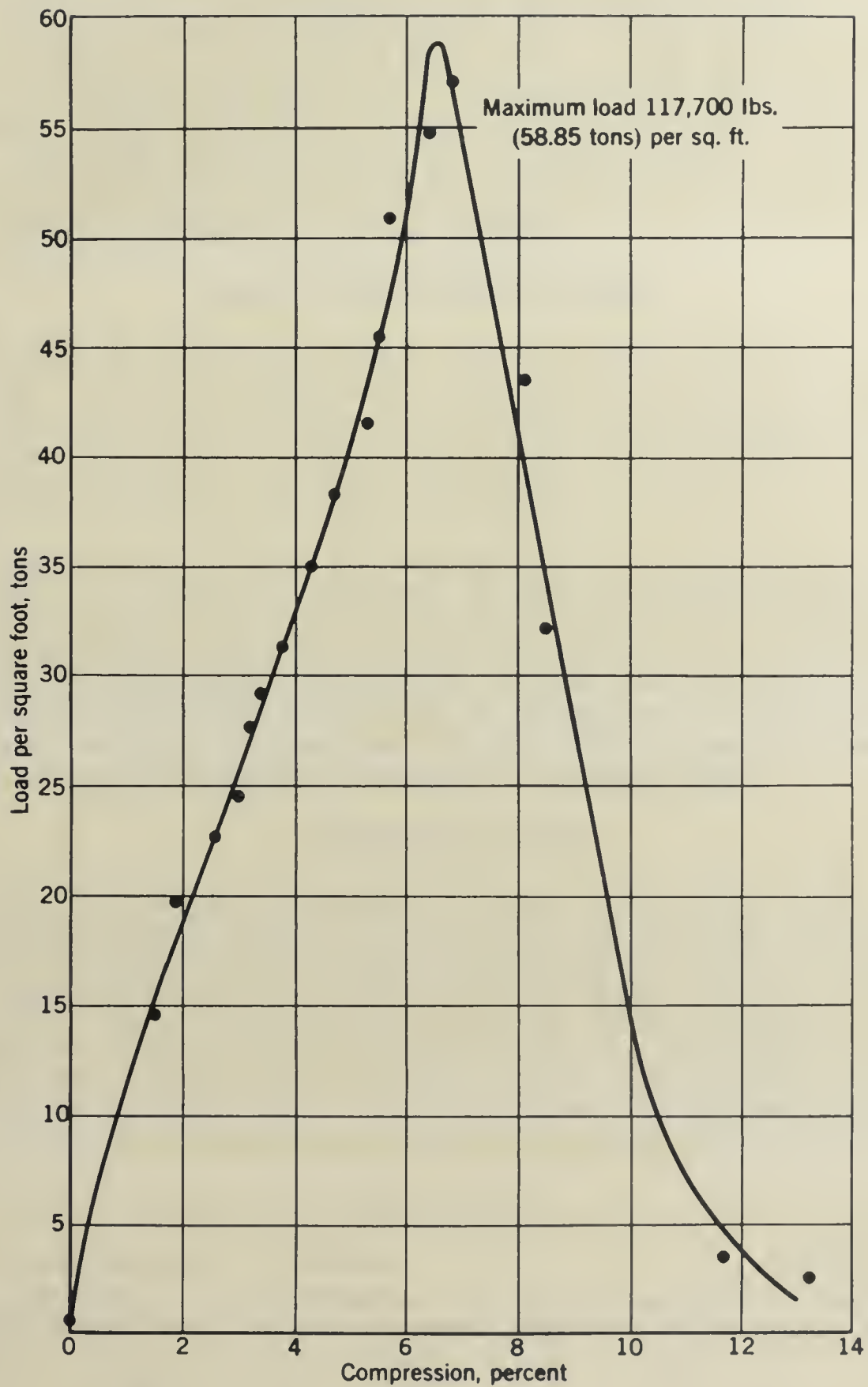


Figure 5.— Relation of compression to load on a 30-inch cube of Pittsburgh coal.

In mine pillars the strength of any small area will depend on the amount of lateral support given thereto. With pillars having a least lateral dimension 5 or more times the height, it seems quite possible that the compressive strength of the center of the pillar will approach the figure obtained with small cubes, namely, 160 tons per square foot. On the other hand, fenders usually have a lateral dimension no greater than their height, if as great, and are consequently much weaker than large pillars. It appears necessary to apply the data of figure 5 to such fenders until better values are obtained.

TESTS OF THE STRENGTH OF POSTS

In order to understand the part played by timber supports, compression tests were made of eight posts. The posts were selected from the stock pile at the mine and were as representative of the supply on hand as such a number could be. There was obtained, also, a supply of sawed cap pieces and wedges. All posts were sawed to a length of 66 inches before testing and the ends were made parallel.

In placing the posts in the testing machine care was taken to reduce eccentricity of loading to a minimum, but this was difficult with the more crooked posts. The testing machine was operated at a speed that reduced the distance between the two heads at the rate of 0.9 inch per hour. This was the slowest speed obtainable. In some tests the ends of a post were directly against the steel heads of the machine, in others there was a cap piece on top of the post, and in still others a cap piece was used beneath the post in addition to simulate a yielding floor.

Description of Posts

There were two oak, three wild cherry, and three maple posts. The three species were designated as O, WC, and M, respectively, and the posts of each species were numbered. Apparently, all posts had been cut from second growth; they were unseasoned and had the bark on. Posts O-1, O-2, and WC-1 were rather small and posts O-1, WC-1, and WC-2 were the most crooked. The smaller posts may have been cut from limbs of trees; the large ones evidently were from tree trunks. All the maple posts were split and had one flat face. Circumferences and crookedness of each post were measured at six places, and the following table summarizes the dimensions:

Data on posts

Post	Circumference, inches		Diameter, inches		Area, sq. in.		Maximum deviation of center line (crookedness), inches
	Minimum	Maximum	Minimum	Maximum	Minimum	Maximum	
O-1	12	15	3.75	4.75	11	18	1.1
O-2	13	14.25	4	4.5	13	16	.5
WC-1	12	16.25	3.75	5.25	11	21	1.4
WC-2	18	20.25	5.75	6.5	26	33	1.2
WC-3	17.25	19	5.5	6	24	29	.7
M-1	18.5	21.5	7	8.5	20	27	.4
M-2	17	20	6	7	16	23	.2
M-3	15.5	18	6	6.75	14	19	.7

Results of Tests

The test data are given in figure 6. The unseasoned condition of the wood did not permit sudden rupture; instead, the posts bent at some point of weakness, such as a knot or sudden change in area. Posts WC-1, WC-2, and M-2 were each compressed nearly 2 inches without complete rupture, but at the same time the weak point in each post deflected laterally 4 to 4.75 inches and the load then carried was only a small fraction of the maximum strength of the post. With strong posts the caps failed first, as illustrated by posts WC-3, M-1, and M-3. The caps used with M-3 were particularly poor and were removed one at a time. The procedure can be followed by the arrows on the curve.

Failing strength of the three kinds of wood, in tons per square inch, was: oak, 0.62 and 0.64; wild cherry, 0.85 to 0.94; and maple, 1.55 to 1.81. The results with oak are not fully representative of that wood, as the specimens were poor. The specimens of each species of wood gave reasonably consistent results on the basis of stress per unit area, but the total strength or supporting power of the individual posts varied widely, as figure 6 shows. The posts alone were compressed 0.2 to 0.4 inch when supporting maximum load. This was 0.3 to 0.6 percent of their original length. Large compression (up to an inch) accompanied by increasing load resulted from failure of cap pieces. There was as wide variation in the strength of cap pieces as in the individual posts.

FIRST SERIES OF UNDERGROUND MEASUREMENTS

Scope of the Work

The first series of underground measurements was made between November 4 and December 5, 1935, in 8 room off 15 butt. Figure 7 is a plan of the area served by 15 and 17 butts in which mining was then in progress. Solid lines show development on November 4, and dotted lines show the additional development to December 5.

There was no information on the magnitude of the convergence records that would be obtained, and it was decided to confine work to a single room and the adjacent butt entries. Locations of 16 points used for convergence measurements are shown on figure 7. They were in pairs, one on each side of the respective passageways and as close to the coal rib as was possible. The individual locations were designated by combinations of numbers and letters, such as 8R-75E or 16B-575N. The first half of the combination designates the passageway and the second gives the distance along it together with the side on which the point was located. The pairs of points were, respectively, at 16B-575, 15B-587, 8R-25, 8R-79, 8R-125, 8R-175 and 8R-225. There were two additional points in the ventilation crosscut to 7 room. Points in 8 room and in the crosscut were installed as rapidly as advance of the room and crosscut permitted.

Over the period of the investigation there were 21 working days in 31 days elapsed time. Full schedule for this period would have been 22 working

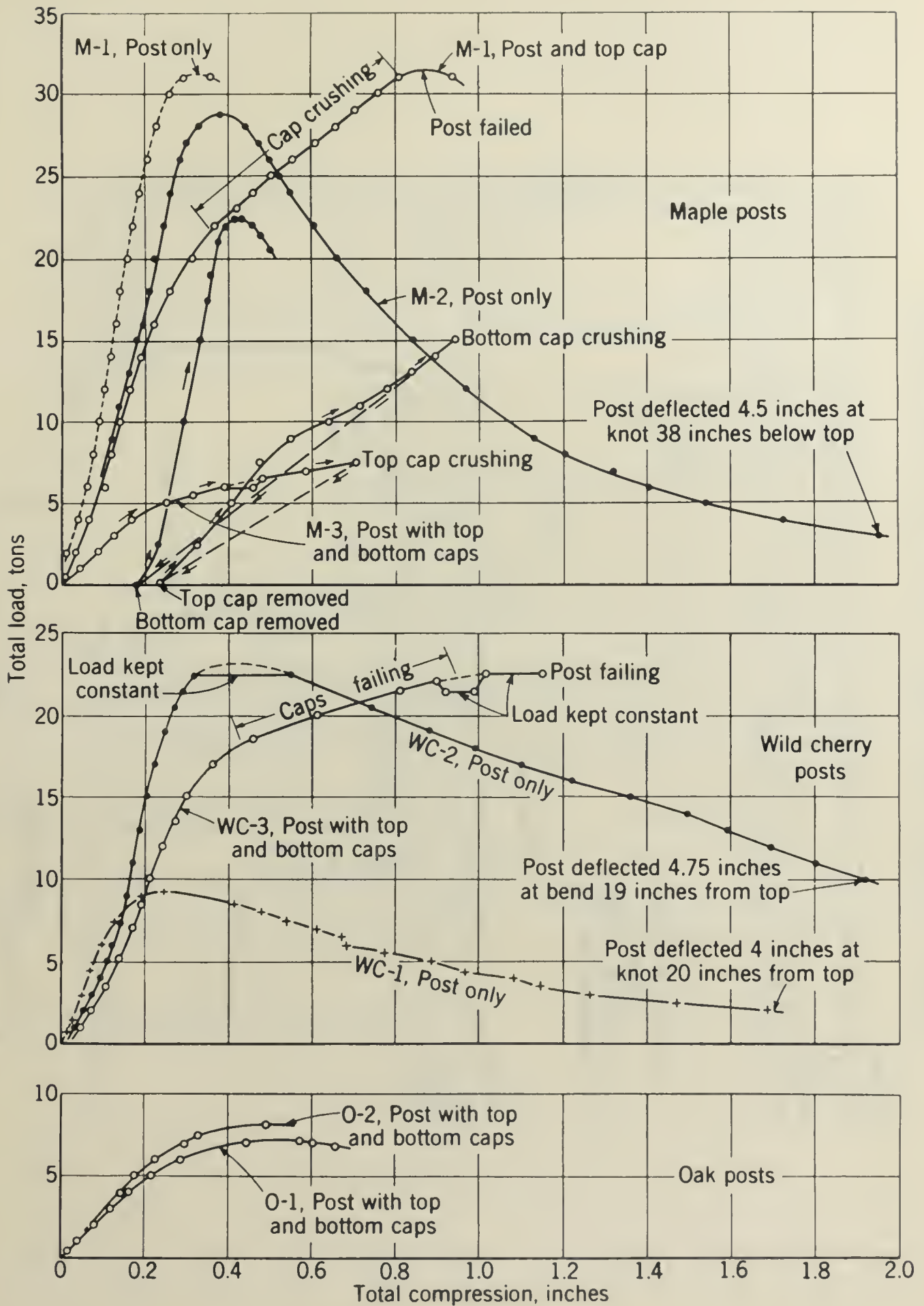


Figure 6.—Data on tests of posts.

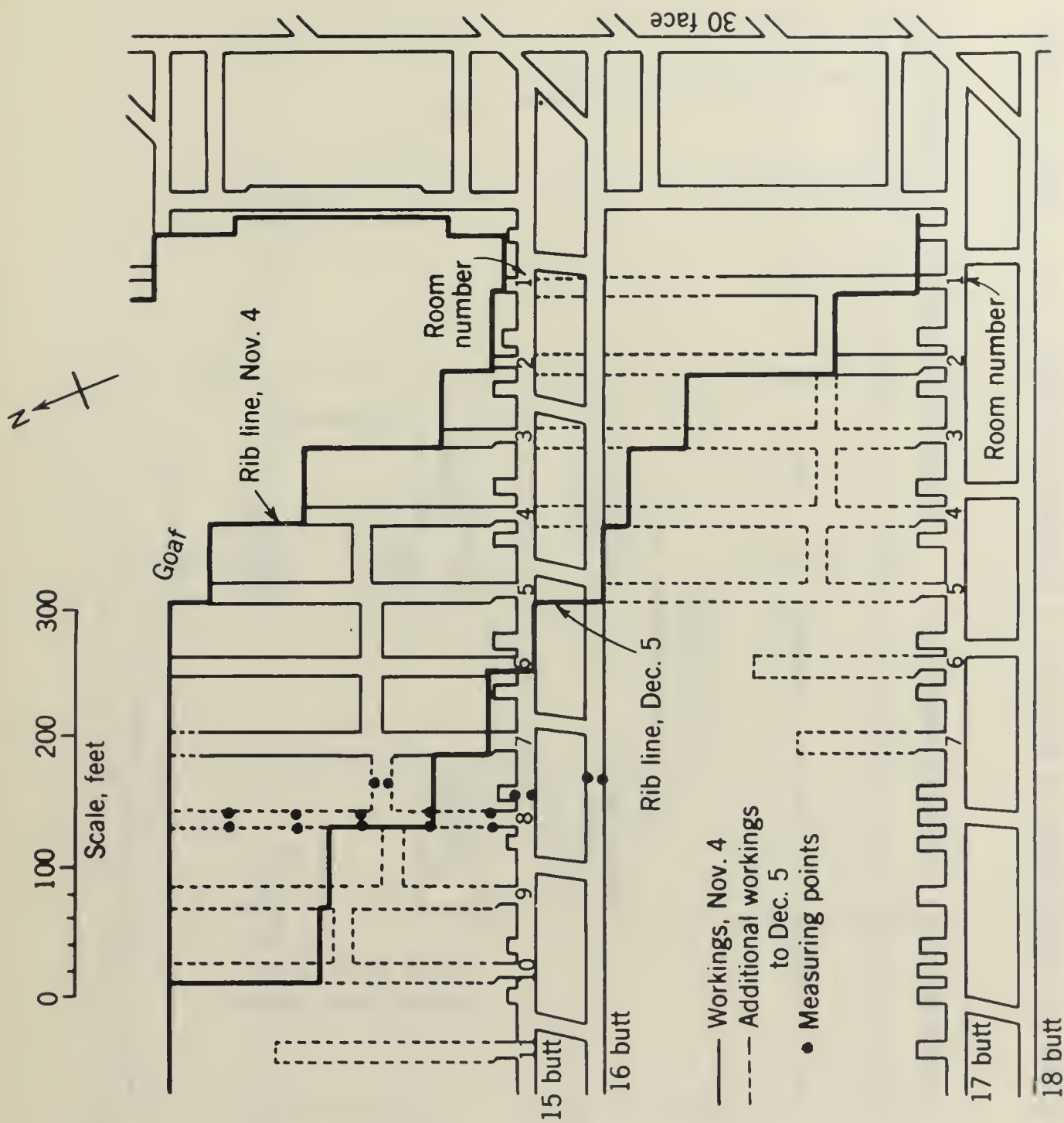


Figure 7.— Details of area studied in first series of measurements.

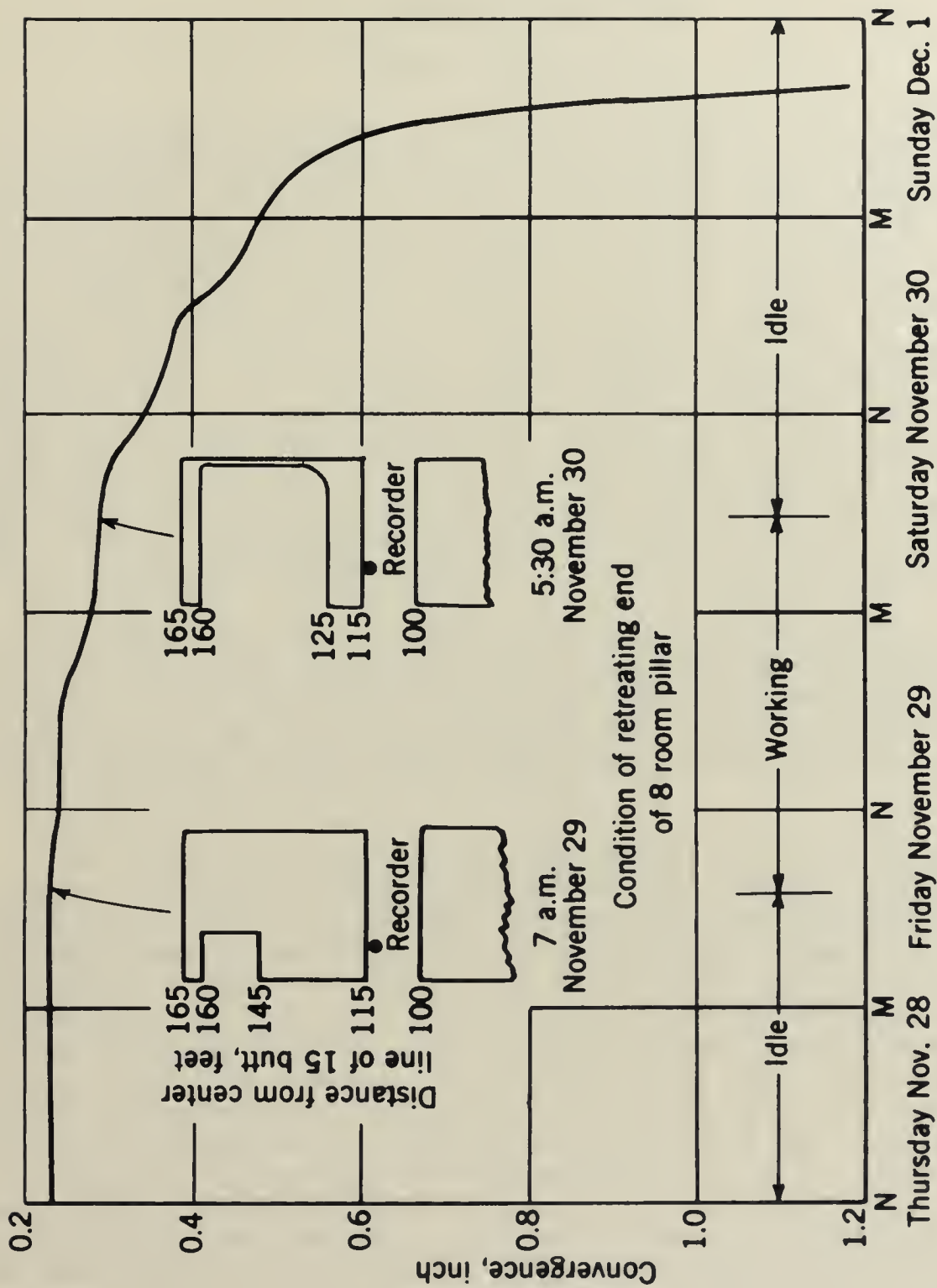


Figure 8.—Convergence at north rib of ventilation crosscut, 8 to 7 rooms, 15 butt.

days. Each working day consisted of three shifts beginning at 7:00 a.m. and ending at 5:30 on the following morning, there being an interval of one hour between end of the first and start of the second shift.

Convergence Records

Data from Recorders

A recorder was placed at 15B-587N on November 5 and a continuous record obtained to December 5. First convergence was recorded on November 12 and amounted to only 0.01 inch. Convergence then continued very slowly and did not reach 0.05 inch until November 20. At this time the distance from the recorder to the rib line in 7 room was 135 feet and in 8 room 215 feet. Gradual convergence continued to a total of 0.08 inch at 5 a.m. November 23, when there was a sudden movement of 0.06 inch followed by another of 0.04 inch at 11 a.m., which brought the total to 0.18 inch. Idle time started at 5:30 a.m. on this date. Distance to the rib line was then 75 feet in 7 room and 165 feet in 8 room. Convergence then continued slowly and reached 0.30 inch at noon of December 5 when the instrument was taken out. There then remained only 25 feet of coal in the pillar being extracted from 7 room and the distance to the rib line in 8 room was 75 feet. The total convergence was less than had been anticipated and the sudden movements probably were confined to the immediate roof.

A recorder was placed close to the north rib of the ventilation cross-cut from 8 to 7 room on November 20 and was continued there till 8 a.m. December 1. Rod measurements had shown 0.11 inch convergence at this point up to the time the recorder was installed and this increased slowly and regularly to 0.23 inch by noon November 28 at which time the retreating face of the pillar was about 55 feet distant.

The record obtained from noon November 28 to 8 a.m. December 1 is reproduced in figure 8 together with two sketches showing condition of the pillar at 7 a.m. Friday, November 29, and 5:30 a.m. Saturday, November 30. Idle days preceded and followed this working day. Figure 8 shows that the roof in the ventilation crosscut was disturbed by removal of coal from the pillar inby. A slight wave of convergence took place shortly after work started on November 29 and a much larger wave was recorded beginning about 6 p.m. Conditions were not in equilibrium when work stopped on Saturday morning and convergence continued in waves, each one larger than the preceding. On Sunday morning Feick found the roof in the crosscut in immediate danger of collapse and took the recorder out only a short time before a heavy fall of the immediate roof occurred despite the presence of timber. No similar fall occurred in the worked-out area from 8R-125 to 8R-160 which was also standing on posts. The large convergence shown in figure 8 probably was confined to the immediate roof. However, it is likely that it was initiated by movement of the Pittsburgh sandstone. Subsidence of this rock would force the immediate roof down on the remaining coal and timbers. The immediate roof strata are so weak that any marked twisting action causes them to break and fall between the timber sets. In the extreme, the sets are over-

thrown. Falls of this nature not followed by breaking of the Pittsburgh sandstone were observed a number of times during the investigation.

Relation to Distance from Rib Line

The data just discussed show that in this case convergence was influenced primarily by distance from the rib line and secondarily by time. When a rib line is stepped, question may be raised as to how distance from any point to it should be measured. In the present case measurement might be parallel or perpendicular to the room or at some intermediate angle. Trial of six methods showed that convergence at different points could be correlated best with distance along the room to the end of the retreating pillar worked from that room. While a crosscut is being driven behind a fender, the rib line is assumed to move from the inby side of the fender to the outby side of the crosscut in proportion to the percentage completion thereof.

There are plotted in figure 9 the records obtained at 8R-79, 8R-125, 8R-175, 8R-225, and in the crosscut to 7 room. With the exception of 8R-225, initial records were obtained while the room was advancing and there was convergence up to 0.06 inch while distance from the rib line (goaf) remained constant. This is a time effect that will be discussed more in detail in connection with the third series of readings. Of the 10 curves on the figure, eight do not show convergence of as much as 0.2 inch until the distance to the rib line was reduced below 70 feet. The exceptions are 8R-79E and the south rib of the crosscut. Apparently these two records were influenced by some local conditions. Really rapid convergence did not begin until the distance to the rib line fell below 40 feet. Convergence at the west rib then lagged considerably behind that at the east rib, as one might expect because the retreating face of the west pillar was considerably behind that of the east pillar.

The records obtained at 8R-25 and on the two butt entries (not shown on figure 9) indicate that the entry chain pillar offers additional support to the roof and that under given conditions convergence may be less at points near the room neck. Apparently the roof over the chain pillar between 15 and 16 butts was tilted toward the goaf, for on November 16 there was as yet no measurable movement at 15B-537S but 0.03 inch divergence at 16B-575N. At the end of the series convergence was 0.09 inch at the former point and 0.02 inch at the latter. More detailed information on the behavior of the roof adjacent to chain pillars comes from the third series of measurements.

Dynamometer Prop Records

Two all-steel dynamometer props were tried out during this series of measurements, but the records were disappointing. One prop gave inconsistent records over the entire period of its use. The other gave consistent records and indicated maximum loads of 20 to 22 tons. However, there is little doubt that these values are higher than the load on surrounding wooden props, which yielded somewhat under load. The effect would be to transfer the load to the steel prop, which could not yield nearly as much.

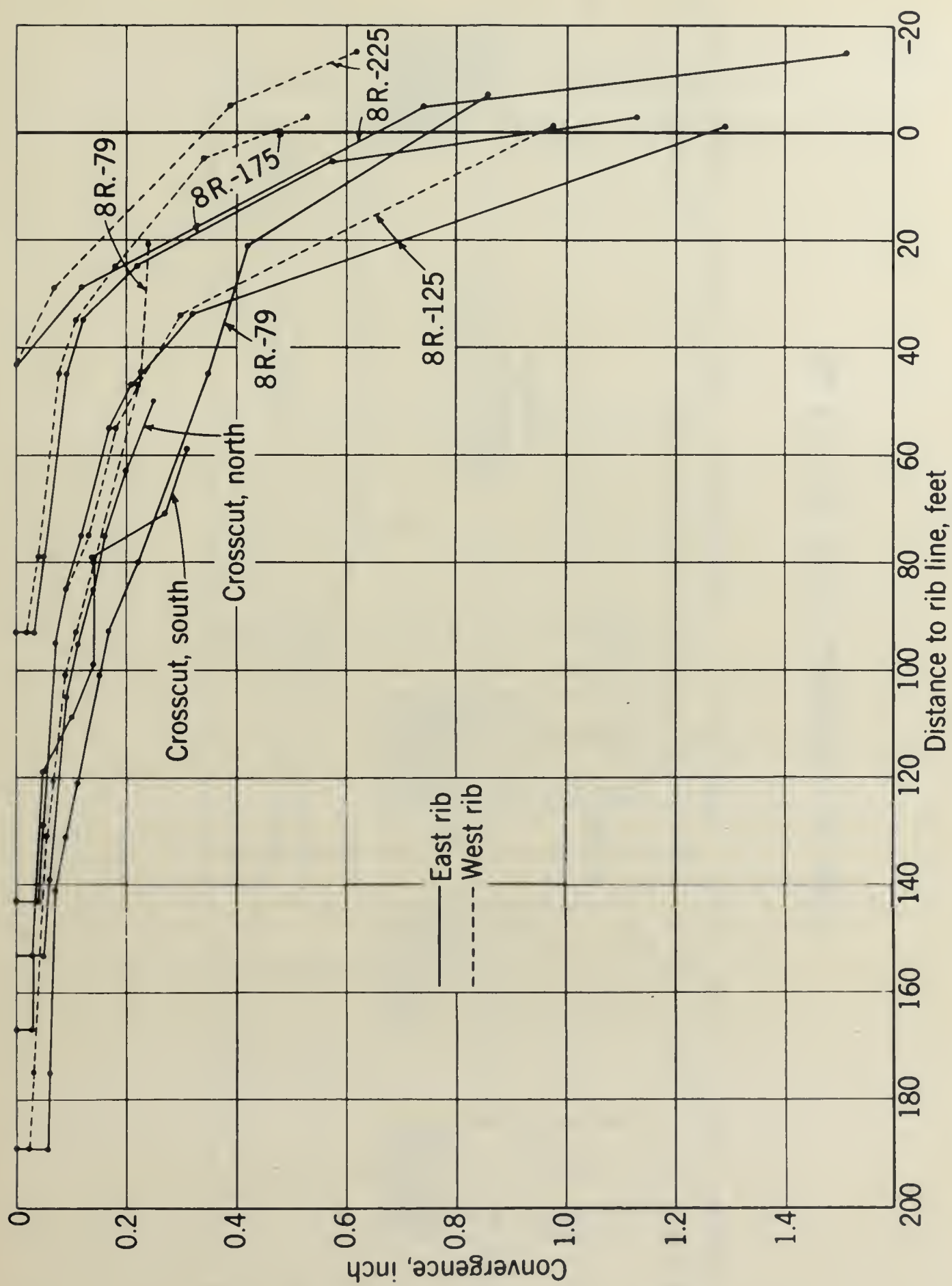


Figure 9.—Relation of convergence of points from 8R.-79 to 8R.-225, 15 butt, to distance from rib line.

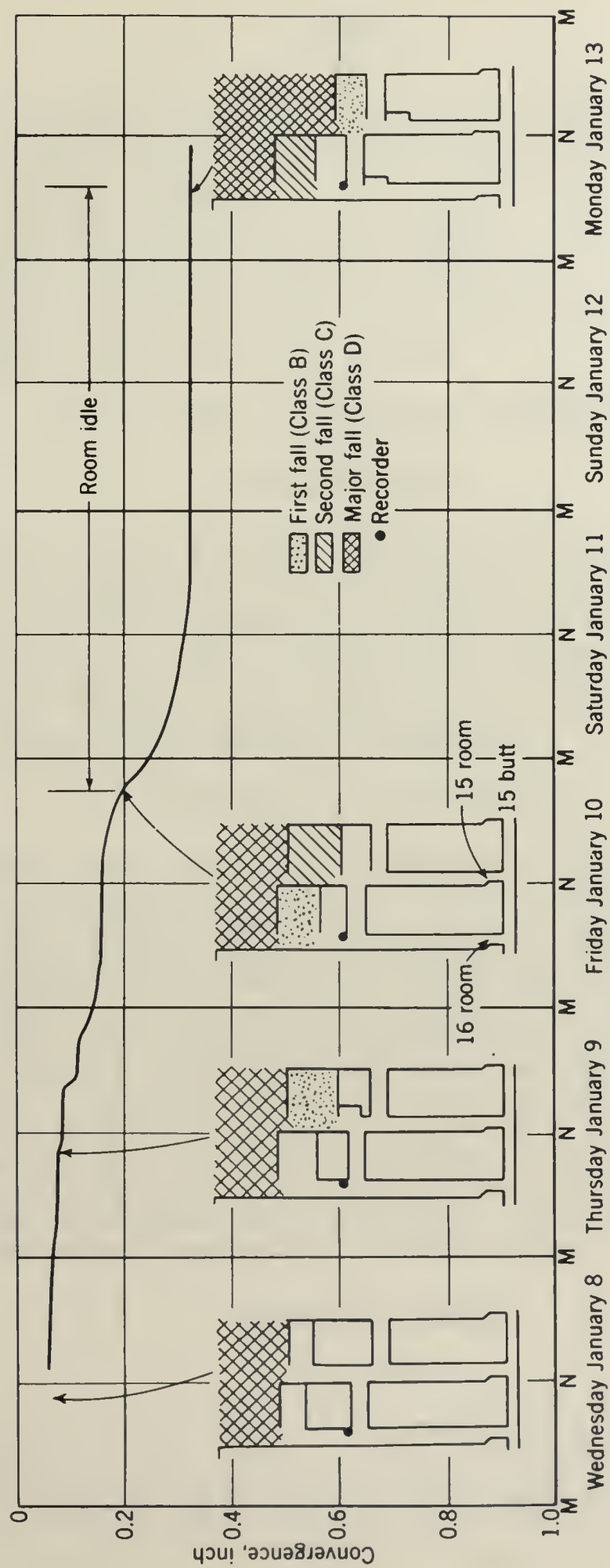


Figure 10.—Convergence at 16R.—150E., 15 butt.

Summary

This first series of measurements, made to obtain a general idea on the behavior of the roof, showed that important convergence was confined to an area certainly within 100 feet and possibly within 50 feet of the face. On the whole, convergence was of less magnitude than had been anticipated; one important record, showing behavior of the immediate roof prior to a fall, was obtained (fig. 8). Distance from the rib line was a primary and time a secondary element in correlation of convergence records. Time effects were negligible in magnitude at points well away from the rib line but of decided importance close to the rib line when work was stopped and an idle period ensued, as figure 8 shows.

SECOND SERIES OF UNDERGROUND MEASUREMENTS

Scope of the Work

In the second series of tests records were obtained along the west rib of 15 room and both ribs of 16 room off 15 butt. Points were placed at 25-foot intervals, beginning at 75 feet and ending at 175 feet from the center line of the butt entry. Readings were begun Friday, January 3, and ended on Saturday, January 13, 1936. Records of the progress of mining in these rooms and of roof behavior in the visible portion of the goaf beyond them were kept, also. The only idle day during the period, in addition to Saturdays and Sundays, was Thursday, January 16.

Designation of Roof Conditions

It was found that correlation of convergence records with roof conditions and extent of falls could be simplified by division of the roof conditions into four classes termed A, B, C, and D. Class A was the normal condition of rooms with the draw slate supported on regular timbering, as previously mentioned. In plotting, this condition is represented by a plain open area. When an area is abandoned and timber is removed, the draw slate and more or less of the roof coal fall immediately. This condition is termed a class B fall and, in plotting, is represented by stippling. Anywhere from 2 or 3 hours to a day or more after the timber has been removed the balance of the immediate roof falls, exposing the Pittsburgh sandstone. This condition is termed a class C fall and, in plotting, is represented by hatching. A major fall does not occur until the Pittsburgh sandstone breaks, which may be several days after timber has been removed. Major falls are termed class D and, in plotting, are represented by cross hatching.

Convergence Records

No. 16 Room

Figure 10 is a reproduction of the chart from a recorder kept at 16R-150E from January 8 to 13. There are plotted, also, plans of the pillars being extracted from 16 and 15 rooms showing extent of mining and nature of falls at the particular times, indicated by arrows connecting each plan with

the curve. The convergence record starts with 0.06 inch, measured with the telescoping rod prior to installation of the recorder. Extraction of coal from 16 room pillar up to a point 25 feet in by the recorder caused no important increase in convergence between 10 a.m. Wednesday and 10 a.m. Thursday. The major or class D fall immediately behind the working places may have been a factor in the case.

A new crosscut was started in 16 room pillar some time after 10 a.m. Thursday, the outby rib of which was close to the recorder. Work in this crosscut doubtless was responsible for the sudden convergence recorded between 4 and 6 p.m. This crosscut was finished by 9 p.m. Friday and there was no further work in the room until 7 a.m. Monday. A wave of convergence that started shortly after noon on Friday was not completed till 24 hours later, after which there was no further movement during the balance of the idle period.

Conditions in 16 room pillar at the cessation of work on Friday evening were somewhat the same as those in 8 room, on which figure 8 is based. The differences are that the rib lines in 16 and 15 rooms were closer together than in 8 and 7 rooms, the area of the goaf standing on posts was less in 16 room than in 8 room, and the class D fall was closer to the rib line of 16 room than to that of 8 room. The net effect of these differences was greater support and less load in the case of 16 room. The result was that the immediate roof in the ventilation crosscut from 16 to 15 rooms did not collapse during the idle period as it had done in the similar crosscut from 8 to 7 room.

No. 15 Room

A recorder was kept at 15R-75W from Sunday, January 12, to Saturday, the 18th, and figure 11 is a reproduction of the record obtained arranged similarly to figure 10. Being at the west rib of the room, this recorder was in position to be influenced by work done in both 15 and 16 rooms. The record starts with 0.10 inch convergence measured by the telescoping rod prior to installation of the recorder. No convergence was recorded until work commenced on Monday morning. A crosscut was then started in 15 room pillar, the outby rib of which was 15 feet from the recorder. Slow convergence started at about the same time and continued during the next 24 hours while the crosscut was being advanced. This crosscut was completed, slab cut taken, and a new crosscut started by 10 a.m. Wednesday morning. It may be that the slab cut was responsible for the sudden convergence at 2 a.m. Wednesday. The amount of coal taken from 15 and 16 room pillars between Wednesday and Thursday mornings was rather small. Convergence continued during that time but ceased shortly after the work was stopped on Thursday morning and did not resume until work started again on Friday morning. During Friday's operations the rib line in 16 room retreated past the recorder, which was then separated from 16 room workings only by a thin fender. Convergence increased rapidly and came in waves of increasing frequency and amplitude. Possibly the sudden break in the curve at 10 p.m. is associated with the final undercutting operations in advance of the crosscut from 16 room. There was a collapse of the immediate roof in the vicinity of the re-

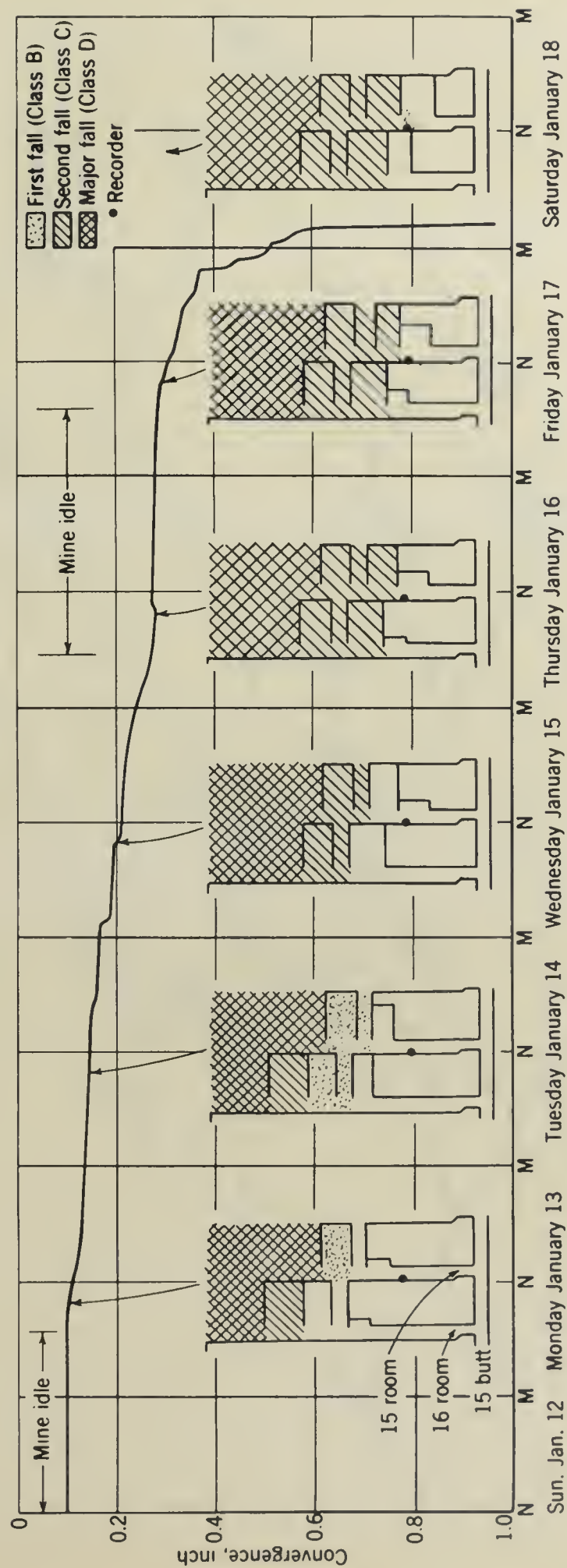


Figure 11.—Convergence at 15R.-75W., 15 butt.

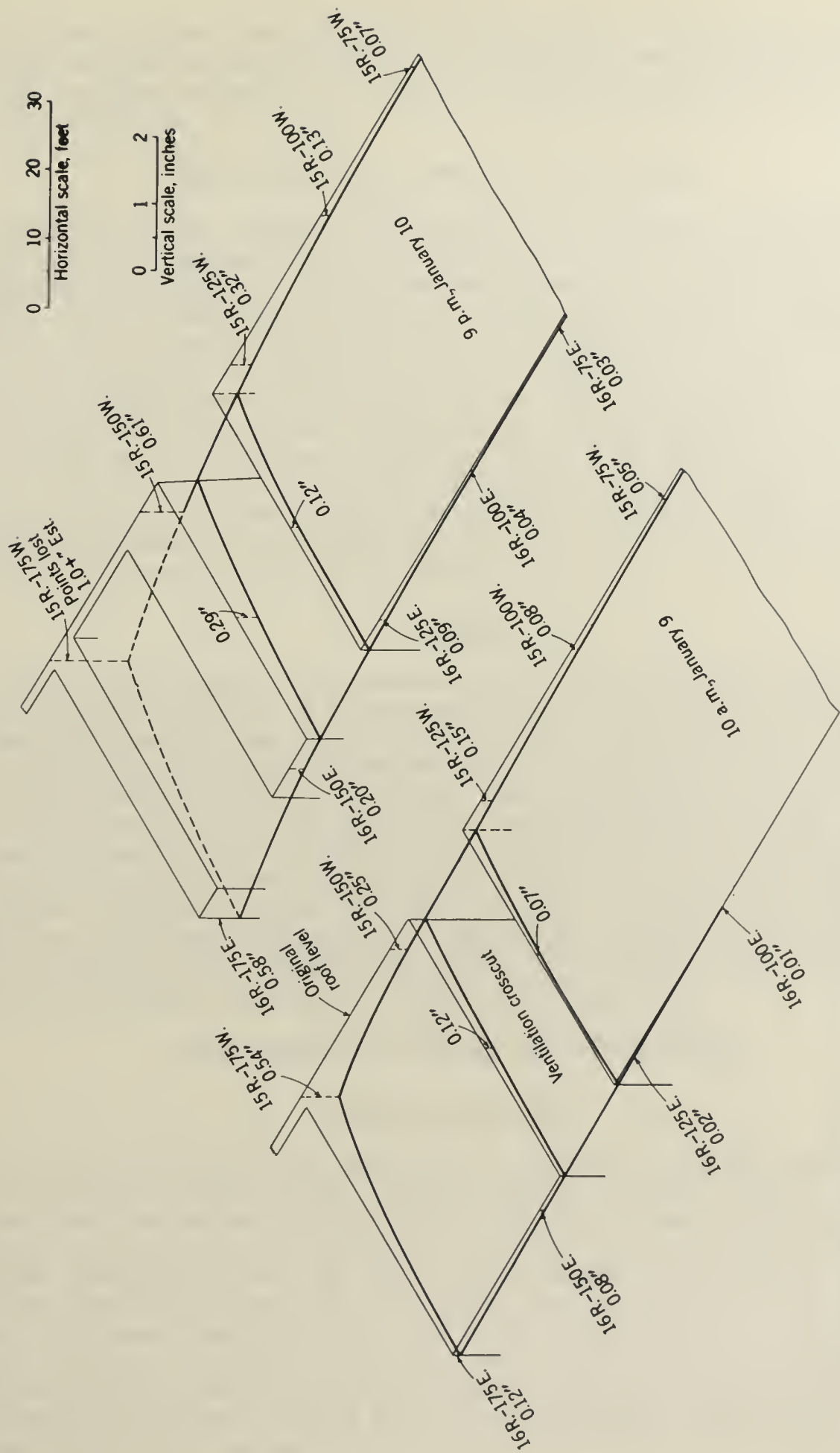


Figure 12.—Convergence across pillar between 16 and 15 rooms, 15 butt at 10 a. m., January 9 and 9 p. m., January 10.

corder shortly after 2 a.m. Saturday. This was a class B fall that carried down timbers and the recorder also. The recorder was dug out Saturday morning, somewhat damaged, but with the clock drum intact and the record perfectly readable.

Three-Dimensional View of Convergence

Figure 12 in an isometric projection to illustrate convergence across 16 room pillar in three dimensions. The two sketches show conditions of mining and convergence at 10 a.m. January 9 and 9 p.m. January 10, respectively. The vertical scale of convergence is 120 times the horizontal distance scale. During this interval a large portion of the block of coal inby the ventilation crosscut was removed. This figure shows that the roof strata were warped toward the corner of the pillar projecting into the goaf at the start of the interval and that this warping increased considerably while mining was in progress. Twisting of the roof strata along the goaf is characteristic of conditions encountered during the investigation. Of course, the amount of twist varies with local conditions. When sufficiently severe, the immediate roof strata are broken so that the timbering can not hold them in place, as has been noted.

Summary

The second series of measurements fully confirmed the first series in showing that convergence of importance was confined to the goaf and to a comparatively narrow zone outby the rib line. Evidently local conditions had considerable influence in determining the exact amount of strata movement associated with working periods. Time effects were evident during idle periods but also depended on local conditions (figs. 10 and 11). The warping and twisting of the roof strata over portions of the pillars projecting into the goaf were demonstrated (fig. 12). It was also possible to show in a limited way the direction of lines of equal convergence across one pillar, but this developed much further in a third series of measurements, as noted below. In general, these first two series of measurements laid the groundwork for a more extended investigation.

THIRD SERIES OF UNDERGROUND MEASUREMENTS

Scope of the Work

It was decided that the third series of underground measurements should cover the entire width, north to south, of the blocks of coal extracted from two consecutive pairs of butt entries and extend across three adjacent room pillars. By the time the second series of measurements was complete insufficient coal was left in the blocks worked from 15 and 17 butts to justify starting the new work in that area, and it was necessary to wait for development of rooms from and establishment of a rib line across 19 and 21 butts. Suitable conditions were reached in the latter part of May 1936 and installation of measuring points started. First convergence measurements were made on June 2 and the underground work continued daily through August 31. During this period the mine worked its regular schedule

of 5 days a week, Monday to Friday, inclusive, with the exception of Friday, June 26, and Tuesday, June 30, on which days it was idle. Measurement of surface subsidence by Pittsburgh Coal Co. engineers started on May 6 and continued weekly from May 18 to October 5, with an additional set of readings on November 2.

Arrangement of Measuring Points

Figure 13 is a plan of the area covered in the third series of measurements. With the exception of the butt entries and 9 and 10 rooms off 19 butt, all development shown on the figure took place during the three months of the investigation. At the start the rib line had not entered this area.

Points for measurement of convergence underground were established at 50-foot intervals in rooms 12 to 15 off 19 butt and 10 to 13 off 21 butt as shown in figure 13. Points were also placed at the middle of each ventilation crosscut between rooms and along the butt entries similarly, the total number being 127. Of course, points could be placed in the rooms only as rapidly as these were advanced. On the other hand, points in the butt entries were in place from the beginning of the investigation until they were lost in the goaf. The purpose was to have as many pillars of coal completely surrounded by measuring points as was possible.

There were 36 surface stations for measuring subsidence, located and numbered as shown on figure 13. Heavy iron pins driven deeply into the ground were suitable leveling points, as there was no danger of frost during the period of the investigation.

Surface stations 1 to 12 and 17 to 28 were directly over convergence points underground as figure 13 shows. Stations 13 to 16 were over the chain pillar between 19 and 20 butts. Stations 29 to 32 were over the south rib of 22 butt and 33 to 36 were 50 feet farther south over solid coal that would not be developed until work in 19 butt was nearly completed. For convenience of reference, the eight rows of surface stations extending north-south are lettered A to H on figure 13.

Thickness of Cover

Figure 14 shows a profile of the surface parallel to the rooms (north to south) through the lines of surface points labeled B and H on figure 13. The original level of all surface stations was known, together with the elevation at a number of points along the entries. In the area studied the coal bed was essentially level north to south and dipped from east to west at about one foot in 60. On figure 14, elevation of the bottom of the coal bed is plotted at 834 feet, which is within one foot of the values for 12 room off both 19 and 21 butts.

The area studied was under a hill, the crest of which was 75 feet south of 20 butt, and over the inner ends of the rooms driven from 21 butt. Pillars of rooms driven from 19 butt retreated under increasing cover and those

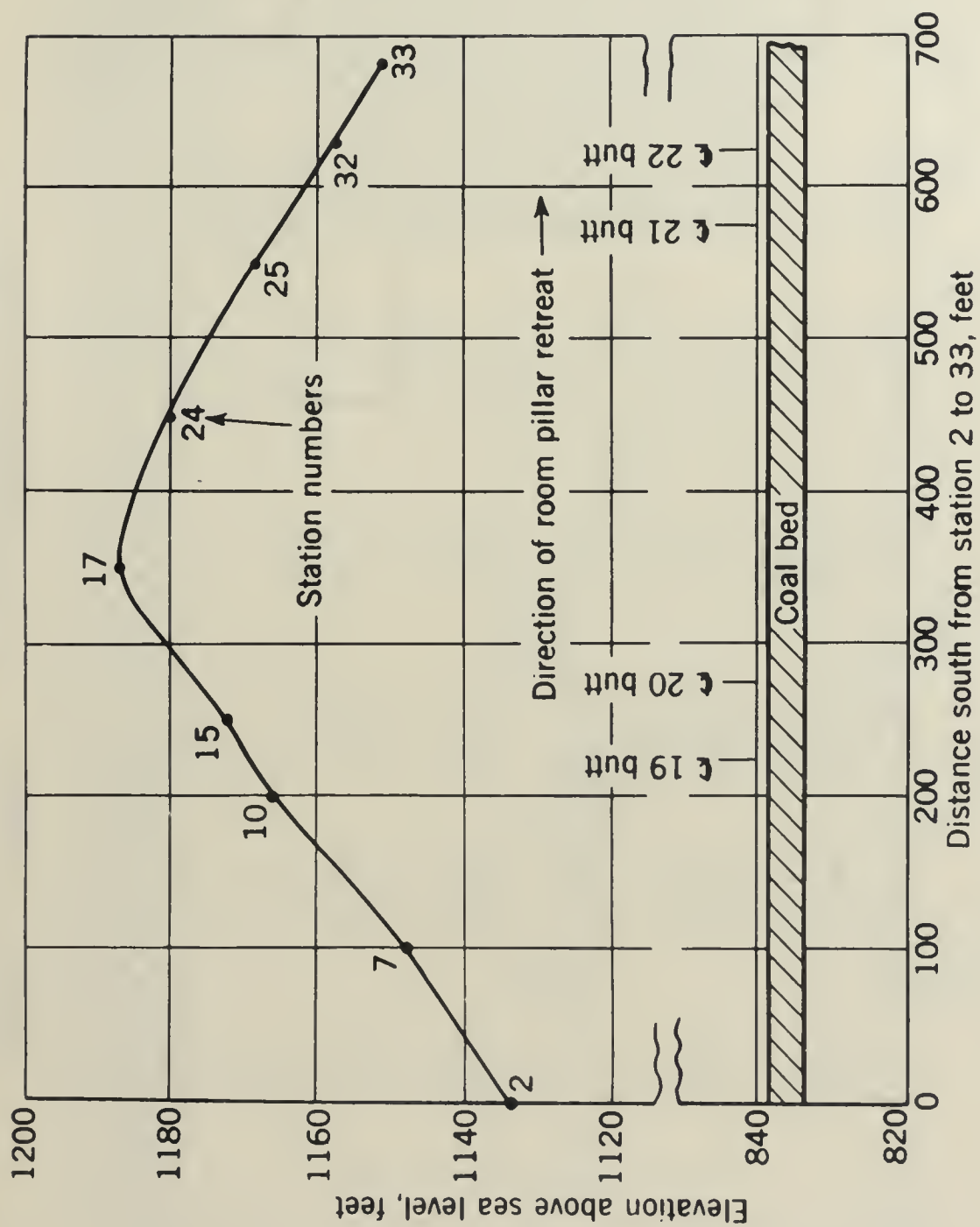


Figure 14.— Profile of surface over area covered in third series of underground measurements.

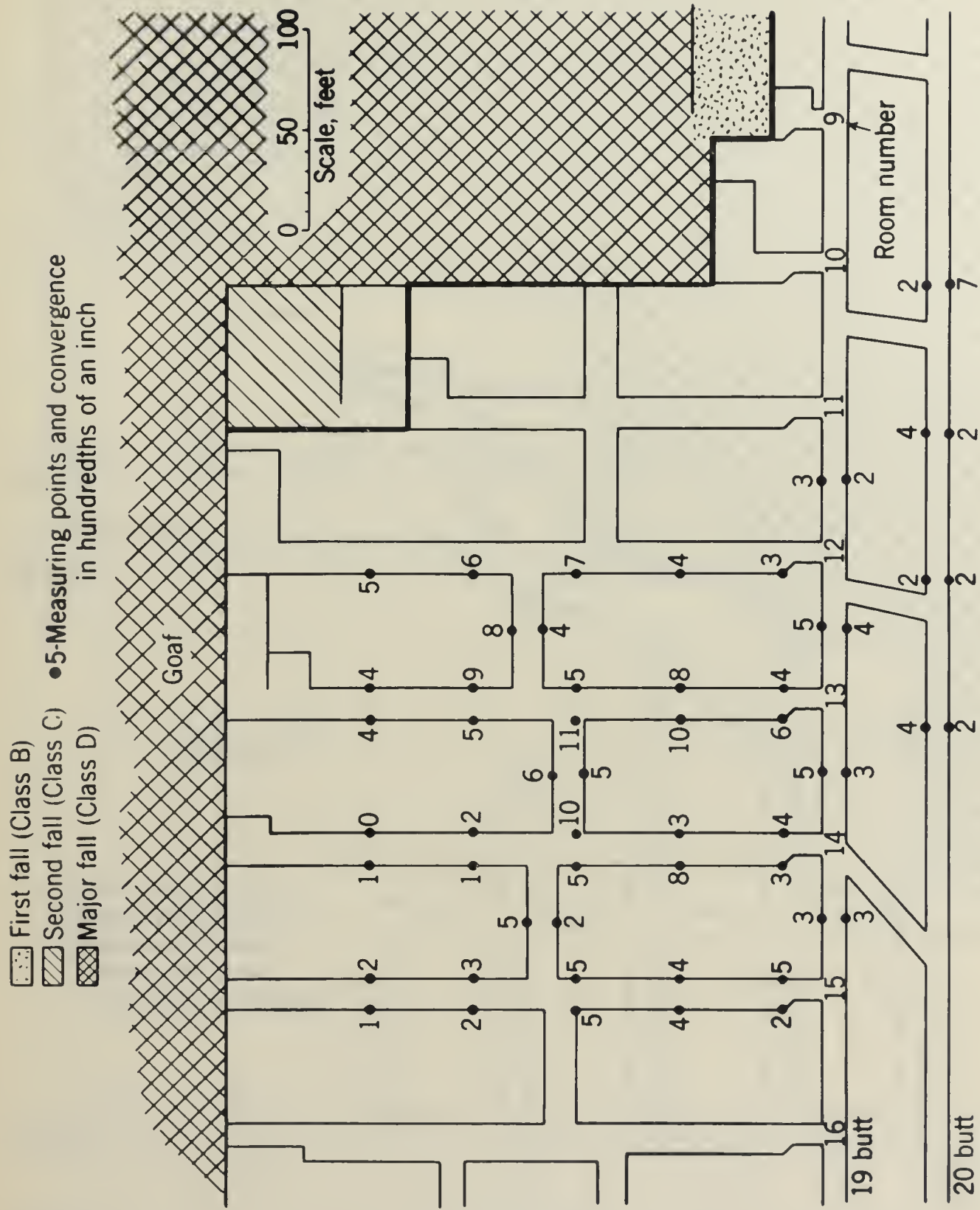


Figure 15.—Convergence and extent of mining and falls in 19 butt area on June 30.

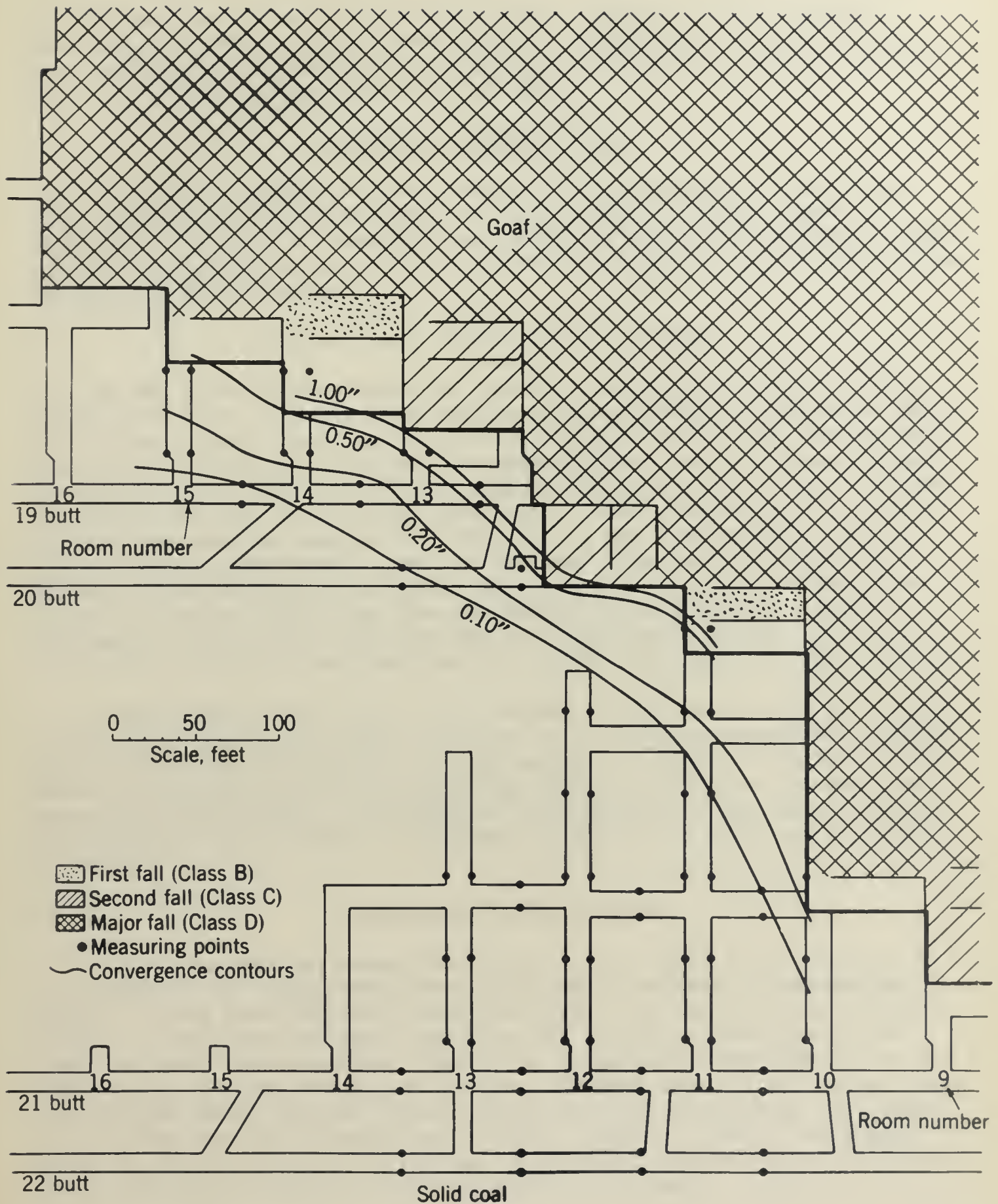


Figure 16.—Convergence and extent of mining and falls in 19 and 21 butt areas, July 25.

from 21 butt mainly under decreasing cover. Contours on the hillside were roughly parallel to the butt entries. Minimum known cover over the coal was 289 feet at station 4 over 15 room off 19 butt. Maximum was 347 feet at station 19 over 11 room off 21 butt.

Convergence Measurements

It is not possible to detail here all the convergence measurements made, as their number is in excess of 3,000. With so large a mass of data, it is possible to determine what might be termed the "reproducibility" of certain elements of the behavior of the strata and to give typical illustrations of that behavior.

Time and Local Effects During Advance

It was observed that minor and somewhat irregular movements of the strata took place as rooms were advanced, and an example is given in figure 15, a plan showing conditions in 19 butt workings on June 30. Points on 19 and 20 butt entries had then been in place 28 days, and convergence ranged irregularly from 0.02 to 0.07 inch with 0.05 inch as second high value and an average for the 16 points of 0.03 inch. Convergence at points 25 feet in the rooms had the same general range as on the butt entries. For all other points in the rooms it can be said that in general the longer the period of installation the greater the convergence. Also, the daily records for any one point show slow and fairly steady movement. There is obviously a time effect here that has no significance insofar as mining is concerned, and local irregularities up to 0.05 inch occur at some places.

The conclusion is that for the conditions here encountered one is not justified in using 0.01 inch as the unit of convergence measurement despite the fact that the instruments are that accurate. Study of all the records show that correlation is facilitated by taking 0.05 inch as the unit, and this is done throughout the balance of this report. As an example, under this system a measurement of 0.12 inch is reported as 0.10, and 0.13 inch as 0.15.

Convergence During Retreat

With a map such as appears in figure 15 as a base, it is possible to plot contours of equal convergence after making suitable interpolations between actual measurements. Interpretation of the data was aided materially by construction of such a map from the records of each day of the third series of measurements. Figure 16, a simplified reproduction of the map for July 25, is given as an example. On this date all pillars of rooms off 19 butt within the area under observation were in retreat. The rooms off 21 butt show both advance and retreat. Major or class D falls were close to the rib line at all points, as there had been extensive falls of this character during the preceding two or three days.

Figure 16 shows that along the rib line convergence was in the neighborhood of one inch. The shape of the one-inch contour was influenced to

some extent by the steps in the rib line, with considerable additional convergence indicated for corners of pillars projecting into the goaf. Evidently there was a relatively sharp bending of the roof strata close to the rib line as the 1.0- and 0.5-inch convergence contours are only 10 to 15 feet apart. This is in contrast to the 0.2- and 0.5-inch contours, which average 30 feet apart in the area north of 20 butt. The 0.10- and 0.20-inch contours indicate quite definite compression of the coal bed ahead of rooms 12 and 13 advancing from 21 butt. The exact location of these contours across the undeveloped area is unknown of course; possibly they should have a reverse curve, as do the 0.5- and 1.0-inch contours above them.

When maps similar to figure 16 are at hand for each day, steady movement of the convergence contours is evident at once and relations with progress of mining and extent of major falls can be traced. Unfortunately, a proper exposition of this would carry the present report beyond the bounds set for it.

Time Effects During Retreat

There were 13 week-end idle periods during the third series of measurements and convergence ranging from 0.10 to 0.51 inch was recorded at 16 points during one or another of them. Analysis of the data shows clearly that movement of the roof strata during idle periods was confined to the goaf and a very narrow band cutting across the pillars close to the rib line. It was here that bending was sharpest and the stresses greatest. Redistribution of stress always lagged considerably behind progress of mining, and during an idle period the redistribution continued toward stable conditions. Sometimes equilibrium was established without complete rupture of one or more of the strata; in others a fall occurred. Figures 10 and 8 illustrate the convergence in these two cases, respectively.

Detailed Study of Cross and Face Cuts

Toward the close of the third series of measurements arrangements were made for detailed observations of convergence, while a crosscut was being driven and face cuts taken in addition. Observers were present continuously during working time to make half-hourly records of convergence and of the progress of mining.

Figure 17 is a line sketch of a portion of the pillar worked from 11 room off 21 butt showing extent of mining at the time of the last detailed measurements. Mining consisted of a crosscut and six face cuts. The areas involved in successive undercutting operations are delineated by dot-dash lines. There were seven measuring points, as shown on figure 17, of which 1 to 6 were placed specially for this phase of the work and 7 was the regular point at 11R-75E. Recorders were used at points 1 and 7 and indicators at points 2 to 6. Because of the rapidity of mining and the desirability of placing points 1 to 6 as soon as possible after the coal was removed, holes were not drilled in the roof for these points. The indicators and the recorder at point 1 were placed under crossbars close to posts; the recorder at point 7 had the usual pins set in drilled holes.

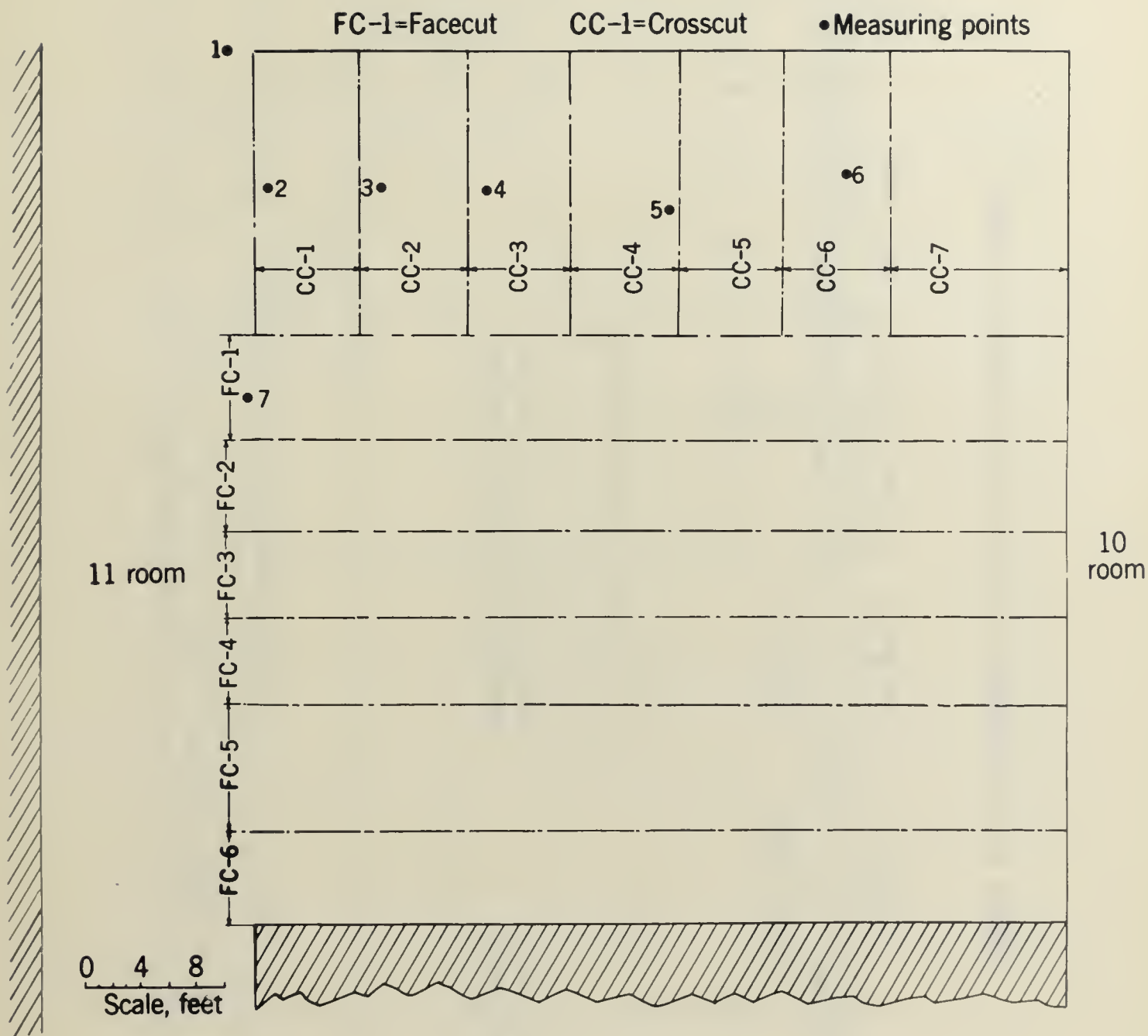


Figure 17.— Plan of portion of pillar in 11 room off 21 butt in which detailed measurements of convergence were made.

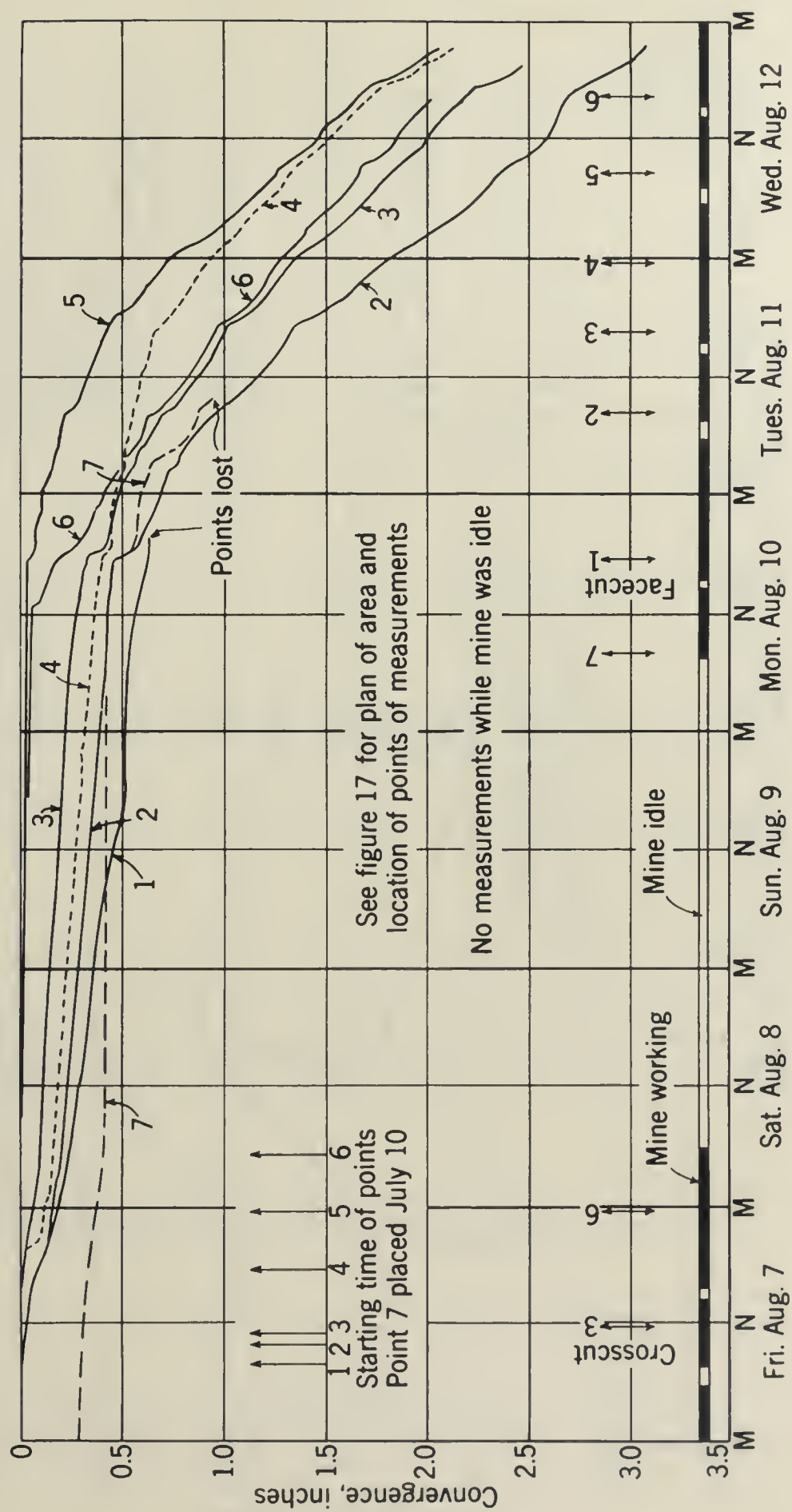


Figure 18.—Convergence records at points shown in figure 17.

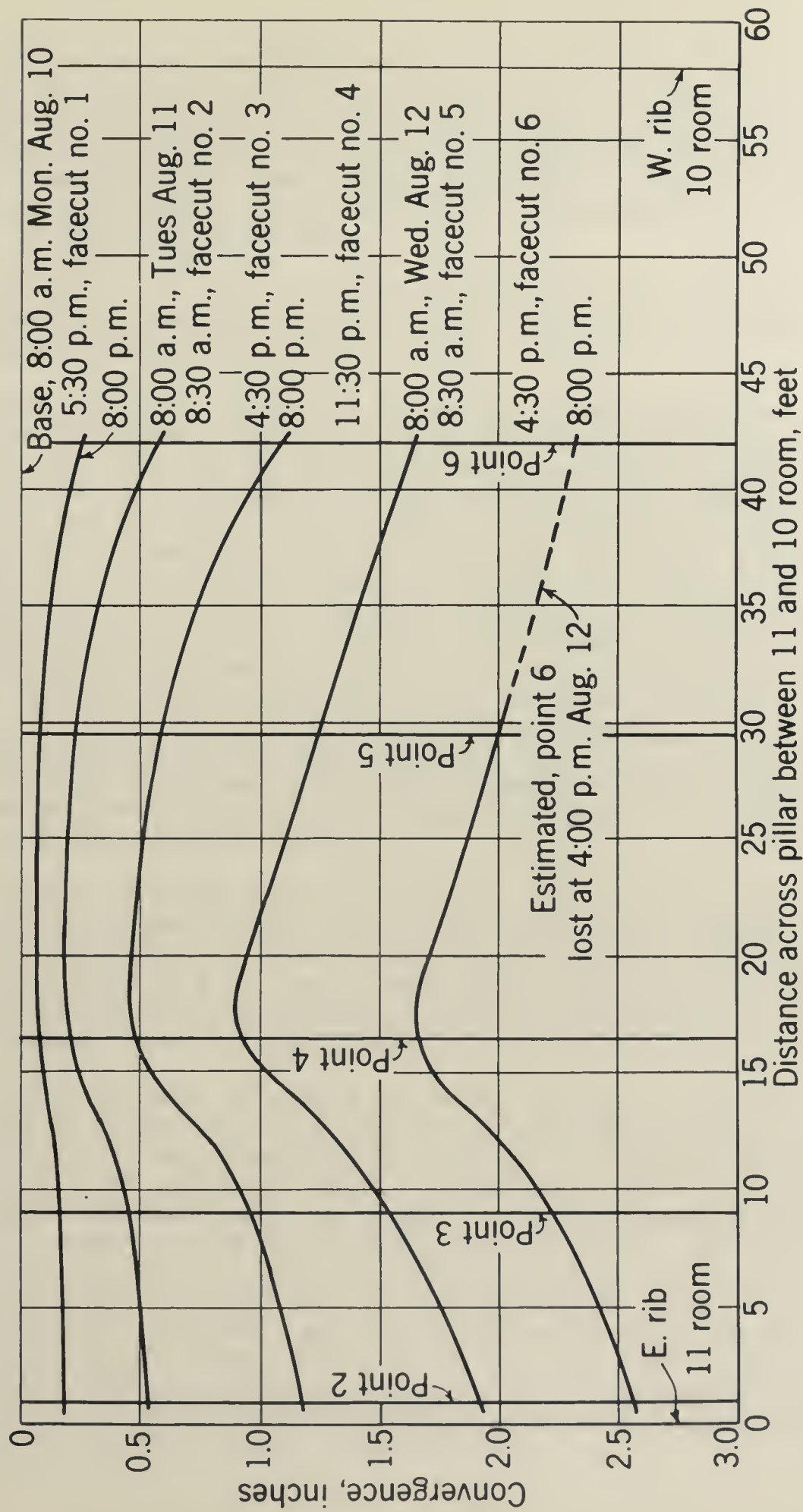


Figure 19.—Convergence profiles along points 2 to 6 of figure 17.

Figure 18 is a reproduction of the records on a time basis. Observations were started on Friday morning, August 7, at which time the crosscut had been undercut twice. The third cut was made just before noon and the sixth was made and cleaned up before quitting time on Saturday morning. Work resumed with the seventh and last undercut of the crosscut on Monday morning, August 10. The first face cut was Monday afternoon and the sixth face cut Wednesday afternoon.

Arrows at the left of figure 18 show the times on Friday and Saturday at which the instruments were placed at points 1 to 6. Records at point 7 had been started on July 10.

Convergence at each point is considered as starting from zero at the time the instrument was placed. Thus, the records obtained at points 1 to 6 all start from zero on figure 18, but convergence of 0.28 inch had been measured at point 7 during the four preceding weeks. There were minor movements of the roof strata up to 5:30 a.m. Saturday, and these continued slowly during the week-end idle period. The curves for points 1 and 7 are exact reproductions of the recorder charts obtained. For points 2 to 6, inclusive, straight lines are drawn connecting convergence readings on Saturday and Monday mornings. Removal of the seventh undercut from the crosscut on Monday morning caused some small additional movement, particularly at point 6. Major movement at all points began with the first face cut on Monday afternoon. Each face cut is marked by a sudden increase in total convergence at nearly all points and in many cases by a subsequent increase in the rate of convergence. The measurements were stopped about 9 p.m. Wednesday, when removal of timber from the area commenced. At that time rapid convergence was in progress at all points.

It must be remembered that convergence at the different points was greater than figure 18 indicates by the amount that occurred prior to placing the instruments. It is impossible, therefore, to tell how much the roof strata over points 2 to 6 were warped from their condition prior to development of rooms. One can, however, select a suitable time as base and show what warping occurred subsequently.

Figure 19 shows warping of the strata during extraction of the face cuts outlined on figure 17. In preparation of this figure, 8 a.m. Monday, August 10, is taken as zero time, and the readings of indicators 2 to 6 at that time are called zero convergence. Any warping prior to this time is neglected. On this basis, the figure shows profiles along points 2 to 6 at 12-hour intervals up to 8 p.m. Wednesday, August 12. The times at which face cuts were made are indicated also. For some unknown reason there was a decided minimum of convergence in the neighborhood of point 4. Initially there was more convergence at point 6 than at point 2, but this condition was reversed 36 hours later. While the profiles all have the same general shape, no two are completely alike in details. As a result of this warping the immediate roof strata were badly broken when the measurements ended, and they collapsed as soon as timber was withdrawn.

After completion of the work in 11 room a similar set of measurements was made in the pillar worked from 12 room, and agreement of the two sets of data was surprising.

Subsidence of Surface

Change with Time

Figure 20 shows subsidence measured on different dates at stations along lines G and A of figure 13. Subsidence is plotted against distance of the various stations north from station 34, and the westerly offset of line A from line G is neglected. The curves represent weekly readings from July 6 to September 14, with a final curve for November 2, seven weeks later. Barring slow changes that might possibly be observed for many months, total subsidence is close to 2.2 feet, or, in round figures, 40 percent of the thickness of the bed extracted. Most rapid movement occurs during the second and third weeks after subsidence starts, after which the rate decreases and at the end of 9 or 10 weeks has become so small that weekly measurements are not justified. In fact, changes at stations 1, 8, 9, and 16 are less than 0.1 foot over the 7-week period from September 14 to November 2.

Study of all the data indicates that rapid subsidence is contingent upon collapse of the Pittsburgh sandstone, and the shape of the initial portions of the curves depends in part on the distance between the rib line and a class D fall.

It is noticeable that total subsidence at stations 31 and 34 was less than 0.2 foot despite the fact that the chain pillar between 21 and 22 butts had retreated far to the west of them on November 2. Observations underground showed that delayed class D falls occurred up to the solid coal over which points 31 to 34 were placed. At the same time, the curve for November 2 (fig. 20) shows that the support given by this coal was greatly retarding if not preventing complete subsidence of the surface at least 200 feet out over the goaf.

Relation to Distance from Rib Line

Figure 21 gives the relation of subsidence at stations 1 to 28, inclusive, to distance of the stations from the rib line at the time the individual subsidence measurements were made. Data from stations 13 to 16 over the chain pillar between 19 and 20 butt entries are marked by crosses and from all others by solid circles. The points scatter considerably and fall in a band, the trend of which is indicated by the dotted central line. This line is a fair average of the scattering individual values. The dash lines connecting points of minimum and of maximum subsidence are used to make the band stand out clearly and have no other significance. Evidently, distance from the rib line is only one of several factors that determine total subsidence at any given point and time. Nor is the relative position of the points the same for all distances from the rib line. For example, stations 1 and 7 had greatest subsidence when 100 to 150 feet from the rib line, but least at 450 to 600 feet.

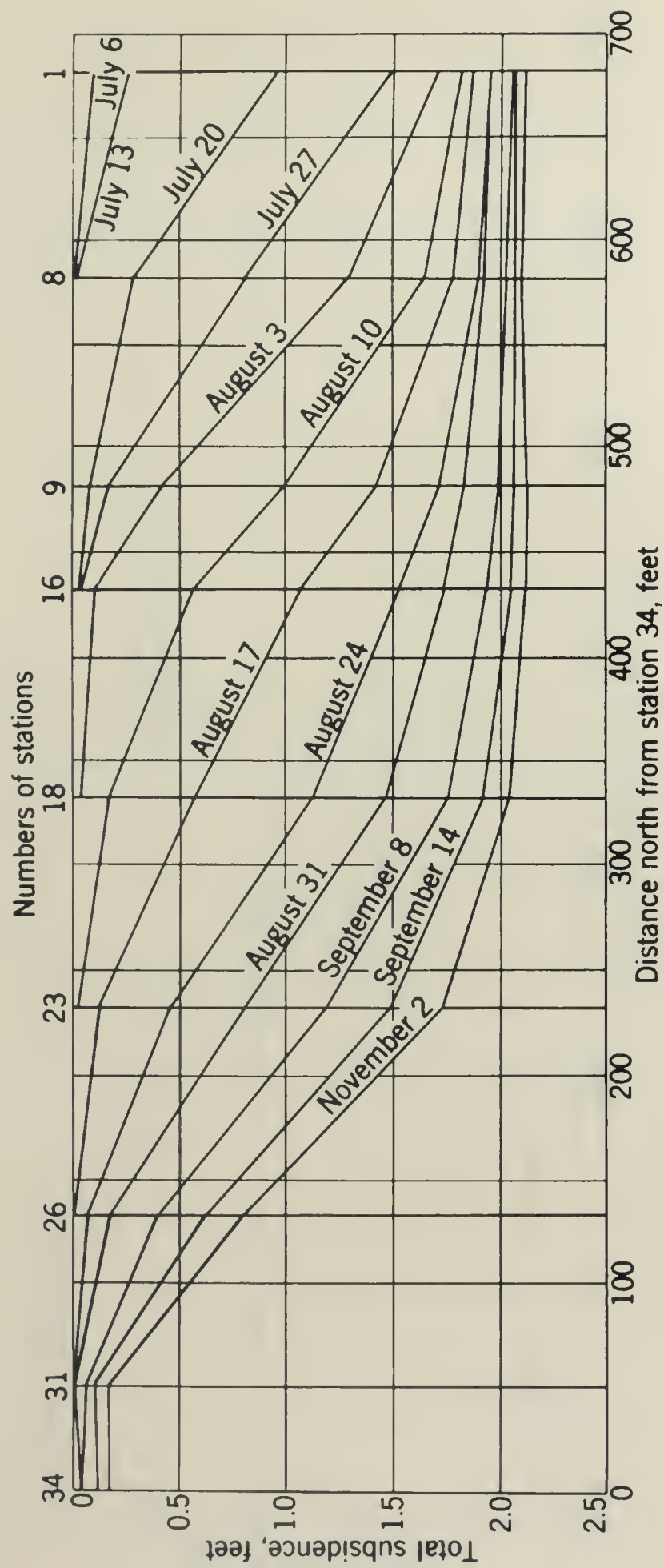


Figure 20.—Subsidence of surface at stations on lines G, and A, July 6 to November 2 (see fig. 13).

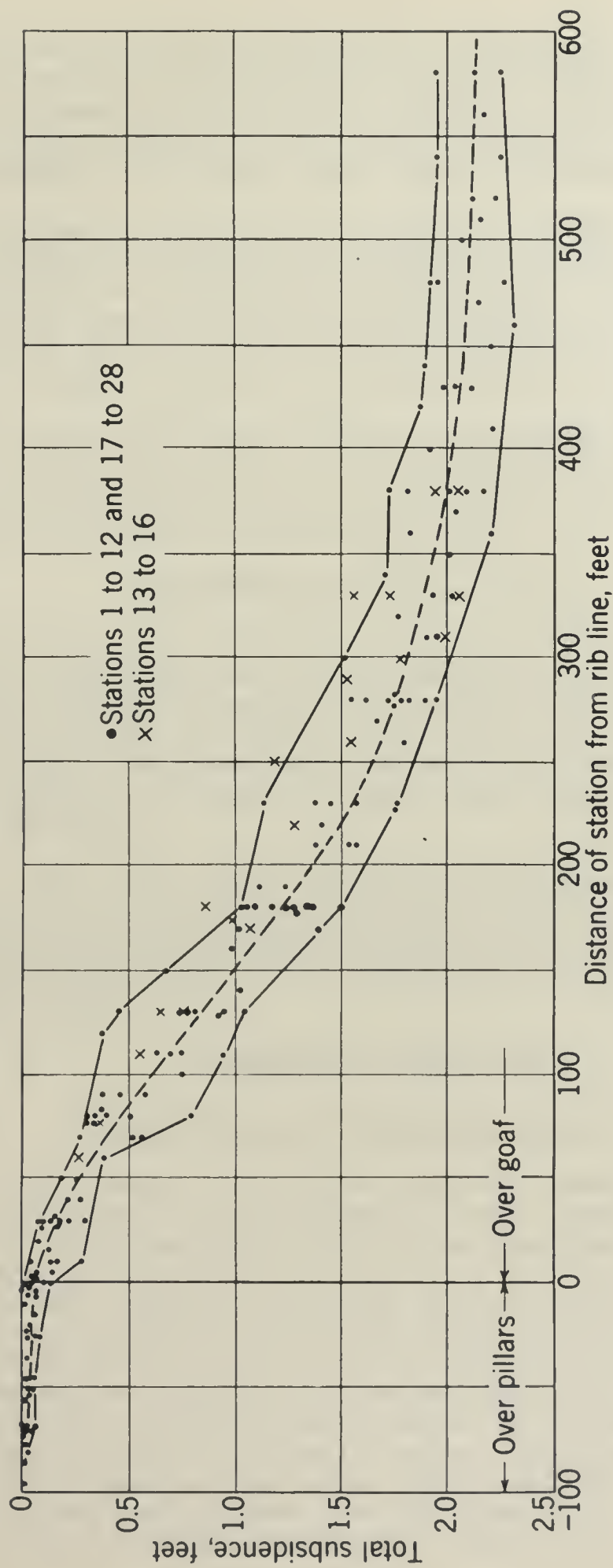


Figure 21.— Relation of subsidence of surface to distance of stations 1 to 28 from rib line.

It should be noted that subsidence at stations 13 to 16 was uniformly less than the average until distance from the rib line exceeded 300 feet. Fenders left in removing the entry chain pillar, combined with the adjacent ends of room pillars, appear to have been particularly effective in lending support to the main roof in the early stages of subsidence.

It is important to note that at this mine subsidence of 0.1 foot or more occurred only over the goaf, in contrast to results published by other investigators in this country and abroad whose data were obtained over slowly moving rib lines or over advancing longwall workings. The strength of the Pittsburgh sandstone and the rate of movement of the rib line are probably the principal causes of the differences in results.

Stations over pillars and in advance of rib lines gave a maximum subsidence of 0.09 foot and that in one case only, with 0.06 foot as second high value. Stations directly over the rib line gave 0.11 foot as maximum subsidence, with 0.06 foot as second high value.

Measurements Over Solid Coal

Stations 29 to 32 were on the edge of the solid coal to the south of 22 butt entry. At the close of the investigation on August 31 extraction of the chain pillar between 21 and 22 butts was half way along the line of these points. Station 29 had then subsided 0.03 foot; the others, nothing. On November 2 subsidence at these four stations ranged from 0.11 to 0.20 foot. Stations 33 to 36, 50 feet back over solid coal, showed subsidence of 0.12 to 0.17 foot on this date.

Retreat of the chain pillar between 21 and 22 butts was followed by a tension crack on the surface directly along the line of points 29 to 32 but approximately 200 feet back of the retreating end of the chain pillar. This crack was about 2 inches wide at the surface and its depth was too great to be ascertained readily.

CORRELATION OF TEST DATA

Significance of Convergence Records

Because of the manner in which the pins were anchored, convergence was measured between a point in the limestone stratum approximately 6 inches below the bottom of the coal and a point in the immediate roof 18 inches above the top of the coal and some 6 or 7 feet below the Pittsburgh sandstone. The floor pins were certainly more solid and less subject to disturbance than the roof pins, and there is no doubt that convergence as measured resulted almost exclusively from movement of the roof pin as long as the point of measurement was on the outby side of the rib line. At the rib line and on the goaf side thereof conditions were somewhat different, as load on the floor was concentrated and applied to smaller areas. The floor in the goaf could be observed only up to the time timber was withdrawn. The thin fireclay immediately under the coal was disturbed easily and breaks in it were common. Some of these were caused by mining operations; others resulted from load on posts.

Floor movements, which with reason might be supposed to involve the limestone, were few and were observed only under extreme conditions of load concentrated on a small area. The error introduced by assuming that convergence resulted entirely from roof movement will be negligible in a first analysis of the data.

Major roof movements depend primarily on the behavior of the Pittsburgh sandstone, and the next question is to what extent the convergence measurements indicate behavior of that stratum. Obviously, at any given point the Pittsburgh sandstone can subside more than the roof pin only if the 6 or 7 feet of intervening strata are compressed. On the other hand, separation of the immediate roof from the sandstone is a decided possibility under some conditions, and this must be taken into consideration. As figures 7 and 13 show, practically all measuring points were originally installed close to pillars; distance from the rib seldom exceeded one foot. The few exceptions were cases in which a point was opposite a ventilation crosscut.

Because of the concentration of roof load around the periphery of a pillar, as a result of bending of the roof strata, separation of those strata at the rib appears to be impossible prior to disintegration of the pillar along the exposed surfaces; nor is it to be expected that very different conditions can exist at points one to two feet from the ribs. It follows that convergence records taken close to ribs of pillars outby the retreating rib line can be used as fair measurements of the movement of the Pittsburgh sandstone.

On the goaf side of the rib line a different set of conditions is found. Here the immediate roof is supported temporarily over large areas only by timbers, and collapses as soon as these timbers are withdrawn. Under such conditions, separation from the Pittsburgh sandstone is possible while the timber is still in place. However, it is not to be expected that such separation will occur to an appreciable extent over fenders unless these are so badly crushed that they have lost their ability to support load. It follows that possibility of separation of the immediate roof from the Pittsburgh sandstone must be kept in mind in assessing the value of all convergence records obtained on the goaf side of the rib line.

One trial was made during the investigation to determine whether separation could be detected in the immediate roof. At 21B-13R-125 a vertical hole was drilled in the roof until the sandstone was encountered. This hole was 7 feet 3 inches deep. It was explored at regular intervals by means of a metal rod having a short and pointed end bent at right angles to the stem. This point found and caught in any cracks that formed between strata. There was a minor separation of the draw slate from the roof coal but no separation at higher points. This hole was opposite a fender and was reached after timber had been removed and a class B fall had occurred. Some 2 feet of the immediate roof then remained unfallen and there was no separation of it from the Pittsburgh sandstone. This may have resulted from the proximity of the fender; conditions were as favorable to prevention of separation as are likely to be found in areas standing on posts.

Support Given By Timbering

Figure 6 shows that there is a wide variation in the strength of posts used. Two poor specimens of oak failed under total loads of 7 and 8 tons, respectively. Six specimens of wild cherry and maple failed under total loads ranging from 23 to 32 tons. In the absence of detailed knowledge of the extent to which the different species of wood and sizes of posts are used underground, an average figure for strength of posts is merely a general estimate.

It is estimated that there was one post for each 11 square feet of roof in rooms, and for each 14 square feet in cross and face cut work. The immediate roof composed of coal and clay slates may be taken to weigh 150 pounds per cubic foot and in the area studied had a maximum thickness of 8 feet. If given no support by fenders and pillars, it would impose a load of 0.6 ton per square foot of roof area, or 6.6 to 8.4 tons per post. On the average, this is possibly one-fourth the strength of the better posts. Obviously, when such posts show signs of heavy stress (as they commonly do in large open areas) the load must result from subsidence of the Pittsburgh sandstone. The total weight of this sandstone and the strata above is 20 to 30 tons per square foot of area, depending on the density and thickness. If it is assumed that 75 percent of this weight is carried by pillars and fenders, the remainder will place a load on the posts two to five times their strength, and in consequence the posts can merely cause a minor alteration in distribution of stress in the sandstone. This statement is supported by the fact that major or class D falls bore little relation to withdrawal of timbers.

The conclusion is that the presence or absence of posts and crossbars has little effect on movement of and stress in the main roof. The function of the timber is to support the immediate roof, and this it does quite well until additional load and distortion of the strata result from subsidence of the Pittsburgh sandstone.

Support Given By Fenders

It is desirable to know what support fenders left in the goaf can give to the roof prior to their being crushed by class D falls. Unfortunately, little can be reasoned concerning this without numerous assumptions that leave the result in considerable doubt. To begin with, knowledge of the crushing strength and compressibility of large blocks of Pittsburgh bed coal is limited to the data given in figure 5, which are admittedly inadequate. Next, it would be necessary to have a fairly accurate measure of the area of the fenders. Some general estimates indicate that fenders might possibly have a supporting power equal to 40 percent of the total weight of the overburden in the present case, but this figure may be rather high. It is certain, however, that the fenders could support considerably more load than the temporary timber; what they actually did support can only be conjectured.

Support Given By Pillars and Gob

The part played by pillars in supporting unbroken roof projecting over the goaf has been discussed previously. The conditions of stress in the unbroken strata can not be analyzed readily without simplifying assumptions. The unbroken Pittsburgh sandstone may be considered as partly restrained between the pillar coal and the superincumbent strata. The coal gives a yielding support that is also irregular because of the stepped rib line and room openings. Warping of the Pittsburgh sandstone must result but probably not to as great an extent as convergence of the immediate roof and floor in the goaf indicates.

The part played by the gob in supporting the roof may be inferred from test data and underground observations. As noted previously, there are many cases in which a class D fall is not in contact with the unbroken portion of the Pittsburgh sandstone for at least half its thickness. In such cases the gob can support only higher strata. In class D falls the Pittsburgh sandstone breaks along the bedding planes as well as perpendicular thereto, and the fallen blocks are seldom, if ever, over 2 feet thick. Under load, the fallen and jumbled mass has a high percentage of voids and yields slowly, as the curves of figures 20 and 21 indicate. These curves and the fact that there are few major cracks in the surface show that in a class D fall the strata do not shear off all the way to the surface. It is certain, then, that some of the weight of strata remaining unfallen over open areas in the goaf is transferred to the gob; how much can not be told. It appears logical to assume that this effect is a minor matter as regards the Pittsburgh sandstone but increases rapidly in importance as the surface is approached.

SUMMARY

The findings that have been discussed in some detail in foregoing sections of this report may be summarized as follows:

1. The nature of the strata overlying the Pittsburgh coal bed in Montour 10 mine is shown by the columnar section given in figure 2. The immediate roof and the Pittsburgh sandstone above it are the strata on which attempts at control should be concentrated.
2. The immediate roof is too weak and friable to support its own weight over spans developed during driving of rooms and extraction of pillars and must be supported on posts and crossbars.
3. The Pittsburgh sandstone is a brittle elastic material with physical properties akin to those found in other types of hard rock, such as limestone. It is variable in composition; two distinct varieties are found in different parts of the mine.
4. Important convergence of roof and floor in the mine is limited to the goaf and a relatively narrow zone along the ends of retreating pillars.

5. In studying convergence, time must be included as a secondary factor, as figures 10 and 11 show. These figures also show some general relations of convergence to different stages in the process of extracting pillars.

6. Measurements of subsidence of the surface show that surface movements over unrecovered pillars are very small. It is probable that this results in part from the rapid movement of the rib line and that different results would be obtained with a slow moving rib line.

7. In conclusion, it may be pointed out that the effect of mechanization and accompanying increase in the rate of movement of the rib line can be determined best by repeating selected parts of the present investigation in a mine in which geologic conditions and the plan of development are essentially the same but in which the rib line moves much more slowly.

APPENDIX

Selected Bibliography

As a part of their work the authors have reviewed literature that deals with theoretical and practical problems of strata movement in mines, subsidence of surface, physical properties of coals and associated rocks, and laboratory tests related to these subjects. All the references found can not be presented within the limits set for this publication; those included are published in English and deal primarily with coal mining. Most of the references are to papers published between January 1930 and October 1936. Brief comments have been added to each reference to indicate the nature of the subject matter.

The bibliography is divided into three sections (lettered A, B, and C) dealing, respectively, with theory and general observations, field data obtained both below and above ground, and laboratory tests of rocks, mine supports, and models. Listing in each section is alphabetical by authors and all items are numbered consecutively to facilitate reference to them. Page references are inclusive of discussion in cases where such discussion took place.

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REPORT OF INVESTIGATIONS

SULPHURIC ACID EXTRACTION METHODS FOR DETERMINING
OLEFINS AND AROMATICS IN HYDROCARBON OILS.
OPTIMUM CONDITIONS AND CONCENTRATIONS OF ACID

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C. H. FISHER AND ABNER EISNER

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SULPHURIC ACID EXTRACTION METHODS FOR DETERMINING
OLEFINS AND AROMATICS IN HYDROCARBON OILS.
OPTIMUM CONDITIONS AND CONCENTRATIONS OF ACID¹

By C. H. Fisher² and Abner Eisner³

INTRODUCTION

Sulphuric acid of various concentrations has long been used as a reagent in the determination of olefins and aromatics in hydrocarbon oils⁴ such as gasoline, kerosene, and neutral oils from coal-tar distillates. Despite its long use, however, there is no general agreement as to optimum conditions and concentration of acid, and numerous methods and modifications have been recommended. In most cases, treatment of the neutral oil with 80 to 90 percent sulphuric acid, followed by distillation to the original endpoint to remove olefin polymers, is recommended for the estimation of unsaturated hydrocarbons. The distillate thus obtained is taken as olefin-free oil and used in later stages of the analysis. Aromatic hydrocarbons generally are determined in the olefin-free oil by suphonation with more concentrated acid (96 to 100 percent and fuming sulphuric acid are recommended), the hydrocarbons not attacked and removed as water-soluble sulphonic acids being considered as saturated hydrocarbons. In some cases the saturates are examined further by differentiating between cyclic (naphthenes) and acyclic hydrocarbons (paraffins). Generally this is achieved by physical methods, although strong sulphuric acid is known to react preferentially with some types of saturated hydrocarbons.

It was the purpose of the present work to make a general study of this type of analysis and to investigate the reactions of hydrocarbons under conditions often used in analyses that employ sulphuric acid reagents. Since previous work is not in good agreement, the optimum concentrations of acid for the different stages of the analysis were determined. Several sulphuric acid methods previously recommended and new modifications were compared, two synthetic solutions being used as the reference oil. The results obtained are presented below under separate headings.

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4 For reviews of this subject see: Howes, D. A., The Chemical Examination of Gasoline and Light Mineral Oils: Jour. Inst. Petrol. Technol., vol. 16, 1930, pp. 54-88; Kestor, E. B., and Pohle, W. D., Determination of Olefin, Aromatic, and Paraffin Hydrocarbons in Neutral Oil from Coal Tar: Ind. Eng. Chem. Anal. Ed., vol. 3, 1931, pp. 294-297; Morrell, J. C., and Egloff, G., Chemical Processes in the Refining of Petroleum Distillates: Nat. Petrol. News., vol. 28, no. 12, 1936, p. 33.

MATERIALS AND PROCEDURE

In most instances the concentrated sulphuric acid solutions were analyzed by titration. The concentration of the more dilute solutions was determined by the specific-gravity method. Usually the hydrocarbons (Eastman's best grade) were used without further purification. Some of the hydrocarbons were purified by washing with sulphuric acid and distillation. A commercial-grade diamylene was distilled through a six-ball Snyder column, only the middle fraction being used. This left in the diamylene an impurity that could be removed by washing with 80-percent sulphuric acid. Bromine-number determinations, as carried out by Francis⁵, were used to ascertain the purity of most of the olefins. Modified separatory funnels⁶ graduated to 20 cc in 0.1-cc subdivisions were used in the sulphonation experiments. It is probable that the accuracy of extractions in this funnel is not much better than 0.1 cc, or 0.5 percent.

Ordinarily the factors of time and relative proportions of reactants were held constant by shaking one volume of the hydrocarbon with three volumes of the acid for 5 minutes. Usually the hydrocarbon was treated with acid at room temperature, any rise in temperature being noted. The mixture was allowed to stand for at least 1 hour before the lower layer was drawn off, and then for at least 1½ hours longer before the volume of unreacted oil was recorded.

Determination of Olefins

Effective Olefin Reagents

It is well known⁷ that, although the more reactive olefins react completely with 80 to 85 percent sulphuric acid, some of the olefins of higher molecular weight (cetene, for example) are hardly affected by acid of this strength. In the present work, a rather unreactive hydrocarbon, diamylene⁸, was used as a standard in the evaluation of various olefin reagents. It has been shown⁸ recently that diamylene is a mixture of the two olefins:



The procedure used in comparing the several olefin reagents was as follows: Fifty cc of diamylene⁹ was shaken with 3 volumes of the acid reagent for 5 minutes at room tempera-

5 Francis, A. W., Estimation of the Unsaturated Content of Petroleum Products: Ind. and Eng. Chem., vol. 18, 1926, pp. 821-822.

6 A.S.T.M. Standards, D114-21T, Barrett-type funnel.

7 Brocks B. T., and Humphrey, I., The Action of Concentrated Sulphuric Acid on Olefins, with Particular Reference to the Refining of Petroleum Distillates: Jour. Am. Chem. Soc., vol. 40, 1918, pp. 825-856; Norris, J. F., and Joubert, J. M., The Polymerization of the Amylenes: Jour. Am. Chem. Soc., vol. 49, 1927, pp. 873-886.

8 Drake, N. L., Kline, G. M. and Roso, W. G., The Diamylenes Produced from Methylisopropylcarbinol by Sulphuric Acid: Jour. Am. Chem. Soc., vol. 56, 1934, pp. 2076-2079.

9 Diamylene that had been purified by distillation and washing with sulphuric acid was not discolored nor was it appreciably attacked by treatment with 80-percent acid. Accordingly, an impurity was mainly responsible for the first loss to the weak acid reagents. The presence of an impurity in the diamylene used to obtain the data in table 1 was indicated also by slight discoloration on the addition of 80-percent sulphuric acid and by a high bromine number. However, the presence of impurities did not impair the usefulness of the material in these comparative experiments.

ture, and the layers were allowed to separate during $\frac{1}{2}$ hour. Except where indicated, the temperature rise was slight. After the volume contraction was noted and traces of sulphuric acid were removed with anhydrous sodium carbonate, the upper layer was distilled through a short Vigreux column. In table 1 the amounts of olefin dissolved by acid and removed on distillation are given. The material boiling over 165° was considered as polymers.

TABLE 1.- Action of olefin reagents on diamylene

Reagent used	Temp., °C., after shaking	Percent olefin removed		
		Dissolved	Boiling over 165°	Total
(1) 80 percent H_2SO_4		10	0	10
Do.		12	0	12
(2) 82 percent H_2SO_4		12.4	8.6	21
(3) 84.5 percent H_2SO_4		8	40	48
(4) 87.5 percent H_2SO_4	39	19.6	74.4	94
(5) 95.5 percent H_2SO_4	43	35	49	84
(6) 80 percent H_2SO_4 + 1 g Ag_2SO_4		16	9	25
(7) 80 percent H_2SO_4 + 2 g Ag_2SO_4		20	8	28
Do.		20	8	28
(8) 82 percent H_2SO_4 + 2 g Ag_2SO_4		20	17	37
Do.				42
(9) 80 percent H_2SO_4 + 1 g Ag_2SO_4 + 1 g Cu_2O		18	6	24
(10) 80 percent H_2SO_4 + 2 g Hg_2SO_4		11	0	11
Do.		12	0	12
(11) 80 percent H_2SO_4 + 2 g Cu_2O		12	0	12
(12) 80 percent H_2SO_4 + 2 g $\text{NiSO}_4 \cdot 6\text{H}_2\text{O}$		11	0	11
(13) 80 percent H_2SO_4 + 2 g FeCl_3		10	0	10
(14) Kattwinkel's reagent	43	26	64.5	90.5
(15) H_2SO_4 + H_3BO_3 + H_2O		19	19	38
Do.		18	10	28
(16) H_2SO_4 + H_3BO_3 + H_2O + 2 g Ag_2SO_4		22	30	52
(17) 85 percent H_3PO_4		9	0	9
(18) 80 percent H_2SO_4 (see footnote).....	48	19	41	60
(19) 82 percent H_2SO_4 (see footnote).....	49	26	31	67
(20) 80 percent H_2SO_4 (see footnote).....	45	26	30	66
(21) 82 percent H_2SO_4 (see footnote).....	33	12	21	33

(14) Kattwinkel's olefin reagent: 100 cc concentrated H_2SO_4 + 20 g H_3BO_3 .

(15) 272 cc 95.5 percent H_2SO_4 + 54 g H_3BO_3 + 42.5 cc H_2O .

(6-13) Grams of inorganic salt added to 100 cc of sulphuric acid.

(16) 2 g. of Ag_2SO_4 added to 100 cc of reagent 15.

(18-19) Used on a mixture (25 cc pine-oil + 25 cc diamylene).

(20) Used on a mixture (25 cc d-limonene + 25 cc diamylene.)

(21) Used on a mixture (20 cc styrene + 30 cc diamylene.)

It will be noted from the data in table 1 that either increasing the concentration of sulphuric acid or adding certain inorganic substances gives reagents of enhanced effectiveness. Silver sulphate was beneficial in all cases, and the sulphuric-boric acid

reagents¹⁰ were better than sulphuric acid alone. The presence of mercury, nickel, ferric, and copper salts was not helpful in the diamylene experiments. The ineffectiveness of many of the reagents, such as 80 to 85 percent sulphuric acid, in dissolving or polymerizing the diamylene completely indicates that the many analytical methods employing dilute sulphuric acid give inaccurate results unless all the olefins present are rather reactive.

In the case of reactive olefins, even moderately dilute sulphuric acid is efficacious. When solutions of d-limonene and pinene in a high-boiling paraffin oil were treated with 80 percent acid under the conditions used above, the undissolved oil was found to distill above the boiling points of the original olefins. To see if unreactive olefins could be removed along with more reactive hydrocarbons by copolymerization, several mixtures containing diamylene and certain reactive olefins were treated as before with three volumes of acid reagent. The results (table 1) indicate that although considerable amounts of olefin had been removed the diamylene was only slightly attacked.

Several experiments were carried out to demonstrate the catalytic effect of silver sulphate and boric acid on olefins other than diamylene. A solution containing 15 percent diisobutylene, 25 percent amylene, and 60 percent diamylene was treated in the manner used previously with three volumes of reagent and distilled. As before, 165° was arbitrarily selected as the distilling temperature separating olefins and their polymers. The superiority of reagents containing silver sulphate was confirmed by these experiments, and the sulphuric-boric acid mixture (reagent 15, table 1, which contains about 80 percent sulphuric acid) was found to be more effective than 80 percent sulphuric acid alone. The most effective olefin reagent, excluding concentrated solutions, appears to be sulphuric acid containing both silver sulphate and boric acid (reagent 16, table 1).

In view of the fact that Houghton and Bowman¹¹ claim that maximum polymerization is obtained with 91 percent sulphuric acid, it is interesting to note that 87.5 percent acid was more efficient than 95.5 percent acid in producing a high-boiling oil from diamylene (table 1). This result should not be interpreted as meaning that 95.5 percent acid is less reactive towards olefins. In the present work the oil not dissolved by 95.5 percent acid started to distill at 120° (instead of 152°, the initial boiling point of diamylene), indicating that another type of reaction (formation of lower molecular weight hydrocarbons) had occurred with concentrated sulphuric acid. It has been shown by Ormandy and Craven¹² that concentrated sulphuric acid converts olefins into a complex product having wide boiling ranges and consisting mainly of paraffins. It is believed that the production of lower-boiling hydrocarbons by concentrated sulphuric acid is responsible for the fact that 87 to 91 percent acid appears to have a greater polymerizing action than 96-percent acid.

Sulphonation with Olefin Reagents

It is obvious that sulphuric acid solutions used to determine olefins must not be concentrated enough to attack aromatic hydrocarbons. To determine the highest concentration of acid that can be used without sulphonating aromatics, the behavior of both pure hydrocarbons and petroleum and coal-tar oils was studied. Toluene, xylene, and mesitylene were used as the pure hydrocarbons because of their well-known ease of sulphonation.

10 R. Kattwinkel, Neue Schwefelsauremethoden für die Bestimmung der Aromatischen und ungesättigten Kohlenwasserstoffe in Benzin: Brennstoff-Chemie, vol. 8, 1927, pp. 353-358.

11 Houghton, C. M., and Bowman, S., The Action of Sulphuric Acid on Cracked Spirit: Jour. Inst. Petrol. Technol., vol. 11, 1925, pp. 583-586.

12 Ormandy, W. R., and Craven, E. C., The Action of Sulphuric Acids on Olefins, etc.: Jour. Soc. Chem. Ind., vol. 47, 1928, pp. 317-20T.

TABLE 2.—Sulphonation with dilute solutions at 25° to 27° C.

Acid reagent	Percent sulphonated or dissolved		
	Toluene	Xylene (mixture)	Mesitylene
(1) 77 percent H_2SO_4		0.0	0.0
(2) 80 percent H_2SO_4	0.12	.25	.0
(3) 82 percent H_2SO_408	.16	.0
(4) 84.5 percent H_2SO_45	.8	.75
(5) 87.5 percent H_2SO_4	2.15	3.58	4.6
(6) $\text{H}_2\text{SO}_4 + \text{H}_3\text{BO}_3$	3.0	7.7	7.7
(7) $\text{H}_2\text{SO}_4 + \text{H}_3\text{BO}_3 + \text{H}_2\text{O}$5	1.0	.0
(8) 85 percent H_3PO_4		3.75	
(9) $\text{H}_3\text{PO}_4 + \text{H}_2\text{SO}_4$	20.5	47.0	95 - 98.0
(10) $\text{H}_3\text{PO}_4 + \text{H}_2\text{SO}_4$		3.0	

(2-5) Average of three trials.

(6) Kattwinkel's reagent: 100 cc of concentrated H_2SO_4 + 20 g H_3BO_3 .

(7) 272 cc 95.5 percent H_2SO_4 + 54 g H_3BO_3 + 42.5 cc H_2O .

(9) 55.5 cc of 85 percent H_3PO_4 + 272 cc of 95.5 percent H_2SO_4 .

(10) 40 cc 95.5 percent H_2SO_4 + 20 cc 85 percent H_3PO_4 .

The aromatic hydrocarbons (20 cc) were shaken vigorously for 5 minutes at room temperature with 3 volumes of acid reagent in a graduated separatory funnel. No attempt was made to control temperature, which increased to about 35° when appreciable sulphonation occurred. The results of these experiments (table 2) show that sulphuric acid of concentration higher than 84 percent cannot be used satisfactorily to determine olefins in the presence of aromatic hydrocarbons.

Kattwinkel¹³ diluted 96 percent sulphuric acid with boric acid and used the resulting solution (which contained about 86 percent sulphuric acid) to determine olefins. As shown in table 2, three volumes of this reagent had an appreciable sulphonating action and therefore can be used with accuracy only under certain limited conditions. Dilution of concentrated sulphuric acid with 85 percent phosphoric acid gives a reagent (80 percent sulphuric acid) still capable of sulphonating some aromatics; but dilution with boric acid and water gives a reagent (80 percent sulphuric acid) that had no appreciable action on toluene, xylene, and mesitylene. These results show that the nature of the diluent has a marked effect on the sulphonating ability of the resulting solution.

It was shown in a more direct manner that 80 percent sulphuric acid has no appreciable action on aromatic hydrocarbons. Four hydrocarbon oils (kerosene and neutral oils from the distillation of low-temperature tar, high-temperature tar, and hydrogenated coal) were agitated in the usual manner with 80 percent sulphuric acid, and an aliquot portion of the aqueous layer thus obtained was treated with barium hydroxide according to the method of Towne¹⁴. By careful neutralization of the excess barium hydroxide with dilute sulphuric acid (phenol-red indicator) and filtration, a filtrate was obtained that would have contained the sulphonic acids as soluble barium salts. When sodium sulphate was added to

¹³ Kattwinkel, R., Work cited in footnote 10.

¹⁴ Towne, C. C., A Modified Absorption-Distillation Method for the Analysis of Cracked Gasolines: Jour. Inst. Petrol. Tech., vol. 17, 1931, pp. 134-141.

the filtrate there was no precipitation of barium sulphate, which indicated that sulphonic acids were absent and that sulphonation had not occurred.

Several sulphonation experiments were carried out with 80 percent sulphuric acid at higher temperatures. Xylene (mixture) and mesitylene (20 cc each) were allowed to run from graduated separatory funnels into Erlenmeyer flasks, the last traces of the hydrocarbons being washed into the flasks with 60 cc each of 80 percent acid. The Erlenmeyer flasks were shaken in a water bath at 41° to 43° C. for 6 minutes and the contents of the flasks were returned to the graduated separatory funnels.¹⁵ The acid layers were run off and the volumes of oil were ascertained in the usual manner. Under these conditions the xylene and mesitylene layers suffered volume losses of 0.75 and 0.25 percent, respectively, indicating that sulphonation had been negligible. Pure m-xylene was treated in a similar manner with sulphuric acid solutions that contained boric acid (54 grams of boric acid in 272 cc. of 95.5 percent sulphuric acid and 42.5 cc of water) and silver sulphate (2 grams of silver sulphate in 100 cc of 82 percent sulphuric acid). The diminutions in volume of the xylene samples were, respectively, 1.75 and 4.0 percent for the boric acid and silver sulphate solutions.

A few experiments were carried out to determine whether the sulphonating action of concentrated sulphuric acid at low temperature is enough to prevent its use as an olefin reagent. One volume of toluene (also xylene and mesitylene) and three volumes of 96-percent acid were cooled to 0° C., mixed, and shaken in an ice bath for 5 minutes. The toluene, xylene, and mesitylene layers lost volume to the extent of 15.5, 27.3, and 50 percent, respectively, and hence concentrated sulphuric acid cannot be used as an olefin reagent under the conditions of these experiments (see also table 8). It is likely that smaller proportions of concentrated acid or slightly lower concentrations could be used at low temperature without sulphonation, but this possibility was not investigated.

The data thus far obtained indicate that, although reagents containing more than about 84 percent sulphuric acid are objectionable, a number of reagents are available which attack olefins energetically without sulphonating appreciable amounts of aromatics. These reagents were examined later to ascertain their effect in promoting condensations between olefins and aromatics (see table 3).

Reactions Between Olefins and Aromatics

It has been shown¹⁶ that fairly concentrated sulphuric acid will cause olefins and aromatics to condense, the products being alkylated aromatic hydrocarbons. In a recent study¹⁷ carried out to determine the effect of acid concentration on reactions of this type, it was found that acid of low concentration caused the formation of alkyl sulphates. In the presence of more concentrated acid, the chief reaction was polymerization of the olefin. Acid of still higher concentration caused condensation between olefins and aromatics. In the case of isobutene and benzene 96 percent acid caused chiefly alkylation, 80 percent acid caused polymerization, and 70 percent acid converted isobutene into ester

¹⁵ It is shown in control experiments that mechanical loss of oil in this procedure is negligible.

¹⁶ Kraemer, G., and Spilker, A., Ueber die Styrol-Verbindungen der Benzolkohlenwasserstoffe und ihr Uebergang in Anthracen bzw. methylierte Anthracene: Ber. Deut. Chem. Gesell., vol. 23, 1890, pp. 3169-3174.

Spilker, A., and Scade, W., Ueber Anlagerungsprodukte von Styrol an aromatische Kohlenwasserstoffe: Ber. Deut. Chem. Gesell., vol. 65B, 1932, pp. 1686-1689.

¹⁷ Ipatieff, V. N., Corson, B. E., and Pines, H., Influence of Sulphuric Acid Concentration upon Reaction between Olefins and Benzene: Jour. Am. Chem. Soc., vol. 58, 1936, pp. 919-922.

without polymerization or alkylation. Propylene and benzene were found to condense in poor yield at 60° in the presence of 80 percent sulphuric acid. These experiments were not carried out under the conditions usually employed in the analysis of hydrocarbon oils and hence cannot be used with satisfaction to predict the behavior of hydrocarbons in analytical procedures.

To find whether aromatics are alkylated under conditions often used in the determination of olefins, several synthetic solutions were shaken for 5 minutes at room temperature with three volumes of acid and then distilled through an efficient column¹⁸ (10 mm diameter and filled with glass spirals¹⁹ for a length of 20 cm), usually with a reflux ratio of 12 to 1. The components of the synthetic oils had boiling points far enough apart to afford easy separation by distillation. The main purpose was to recover the unreacted aromatic quantitatively, and hence no effort was made to isolate and identify the alkylated aromatics and olefin polymers. Blank distillations indicated that this method could be used with confidence to recover unreacted aromatics. The results (table 3) include values also for the volume losses (ester formation) caused by shaking with the acid reagent, which are expressed as percent of the total amount of olefin originally present.

Of the various olefin reagents studied, only 80 percent sulphuric acid failed to cause condensation between olefin and aromatic hydrocarbons²⁰. Since acid of this concentration did not cause alkylation when reactive hydrocarbons (such as pinene, limonene, toluene, and m-xylene) were used, it is believed that from this standpoint 80 percent sulphuric acid is a suitable olefin reagent. Alkylation was caused by 82 percent acid, except when the comparatively unreactive olefins (diamylene and octylene) were used. This result leads to the thought that treatment with 82 percent sulphuric acid, after removal of the more reactive olefins with 80 percent acid, might be advantageous in removing olefins but incapable of causing olefins and aromatics to condense (See table 8). The olefin reagents (containing silver sulphate and boric acid) observed previously (table 1) to be efficacious in attacking diamylene and ineffective in sulphonating aromatics were found to remove considerable amounts of aromatics as high-boiling alkylated derivatives. Therefore, these reagents cannot be employed satisfactorily for the determination of olefins.

It is apparent from the data in table 3 that silver sulphate and boric acid in sulphuric acid solutions catalyze the alkylation of aromatics. It is believed that this observation has not been made previously, although it has been demonstrated²¹ that boron fluoride in sulphuric acid catalyzes the propylation of benzene.

A comparison of the effectiveness of olefin reagents in causing sulphonation (table 2) and alkylation (table 3) of aromatics indicates that the latter reaction is more easily brought about. This points to the conclusion that alkylation rather than sulphonation is the chief cause of loss of aromatics during the determination of olefins. Since reagents incapable of sulphonation may cause considerable alkylation it appears that sulphonic acids are not intermediates in alkylation reactions.^{22 23}

18 Almost identical with a column previously described (Tongberg, C. O., Quiggle, D., and Fenske, M. R., Efficient Small-Scale Fractionating Equipment: Ind. Eng. Chem., vol. 26, 1934, pp. 1213-17).

19 Wilson, C. D., Parker, G. T., and Laughlin, K. C., A New Glass Packing for Laboratory Fractionating Columns: Jour. Am. Chem. Soc. vol. 55, 1933, pp. 2795-2796.

20 Eight synthetic solutions containing one or two olefins (cyclohexene, diamylene, amylene, d-limonene, pinene, octylene, and styrene) with one or two aromatics (benzene, toluene, and m-xylene) were used in these experiments. The aromatics were recovered quantitatively in all cases.

21 Wanderly, H. L., Sowa, F. J., and Nieuland, J. A., The Olefin-Benzene Condensation with the Sulphuric Acid Catalyst: Jour. Am. Chem. Soc., vol. 58, 1936, pp. 1007-1009.

22 Ipatieff, V. N., Corson, B. B., and Pines, H., Work cited (see footnote 17).

23 Wunderly, H. L., Sowa, F. J., and Nieuland, J. A., Work cited (see footnote 21).

TABLE 3.- Olefin reagents on synthetic solutions

Reagent no.	Solution no.	Composition, percent by volume		Fate of constituents, percent by volume	
		Olefins		Olefins dissolved	Aromatic recovered
		Olefins	Aromatic		
1	1	Pinene, 25 Diamylene, 25	Toluene, 50	37	82
1	2	Limonene, 26 Diamylene, 30	Toluene, 44	36	95-100
1	3	Amylene, 25 Diamylene, 25	Toluene, 50		76
1	3				78
1	3				82
1	4	Octylene, 13.3 Diamylene, 33.2	Toluene, 53.5	6.5	100
2	3	Amylene, 25 Diamylene, 25	Toluene, 50	34	66
2	1	Pinene, 25 Diamylene, 25	Toluene, 50	37	68
3	3	Amylene, 25 Diamylene, 25	Toluene, 50	29	50
4	3			34	40
4	1	Pinene, 25 Diamylene, 25	Toluene, 50	43	52
4	5	Diamylene, 50	Toluene, 50	28	60

Reagents: (1) 82 percent H_2SO_4 .
 (2) 80 percent H_2SO_4 + Ag_2SO_4 (2 g added to each 100 cc).
 (3) 272 cc 95.5 percent H_2SO_4 + 54 g boric acid + 42.5 cc water.
 (4) 2 g Ag_2SO_4 added to each 100 cc of reagent 3.

Determination of Aromatics

Effective Aromatic Reagents

A suitable sulphuric acid reagent for removing aromatics must sulphonate all aromatics present without attacking saturated hydrocarbons. In studying this phase of the sulphuric acid method of analyzing oils, several representative aromatic, naphthenic, and paraffinic hydrocarbons were used.

Concentrated sulphuric acid (3 volumes of 95 to 96 percent) was found to sulphonate completely most of the aromatic hydrocarbons studied (toluene, xylene, ethylbenzene, isopropylbenzene, diethylbenzene (mixture of meta and para), p-cymene, pseudocumene, and beta-methylnaphthalene). Under the same conditions, normal, secondary, and tertiary butylbenzenes, respectively, were 76, 74, and 60 percent sulphonated. The hydrocarbons sulphonated with most difficulty were diphenylmethane, tertiary amylbenzene, benzene, and cyclohexylbenzene (14, 29, 35, and 42 percent sulphonated). These results roughly parallel those of Towne²⁴, who employed 93 percent acid as the sulphonating agent. An attempt to study the sulphonation of solid aromatic hydrocarbons of high molecular weight in hydrocarbon solvents was not very satisfactory and is not described. However, it was demonstrated in a later experiment that treatment with 3 volumes of concentrated sulphuric acid removed almost all the aromatics in a tar oil (see table 5) that had a distillation endpoint of 350° C.

A reagent (98 percent sulphuric acid) often recommended for the determination of aromatics was found to be a powerful sulphonating agent. The Kattwinkel sulphonating reagent²⁵ (100 cc of concentrated sulphuric acid containing 30 g of phosphorus pentoxide) also was found to be an efficient sulphonating agent, surpassing even 98 percent sulphuric acid in one instance. These data are shown in table 4, where the results obtained with 95.5 percent sulphuric acid are included for the sake of comparison. The addition of silver sulphate gave reagents of enhanced sulphonating strength. Phosphorus pentoxide also is capable of increasing the sulphonating action of sulphuric acid solutions, but this possibly is due to its absorption of water rather than to any catalytic effect. A study of the action of fuming sulphuric acid on aromatic hydrocarbons was not carried out, since such reagents attack paraffins (see table 6) and hence are not suitable for hydrocarbon analyses.

TABLE 4.- Comparison of sulphonating reagents

Sulphonating reagent	Percent sulphonated				
	Ben- zene	Xylene	Diphenyl- methane	Cyclohexyl- benzene	t-Amyl- benzene
(1) 95.5 percent H ₂ SO ₄	35	96	13.5	42	29
(2) 98 percent H ₂ SO ₄	100	60	100	100
(3) Kattwinkel's reagent	97	97	74
(4) 95.5 percent H ₂ SO ₄ + 1 g Ag ₂ SO ₄	44
(5) 95.5 percent H ₂ SO ₄ + 2 g Ag ₂ SO ₄	43
(6) 95.5 percent H ₂ SO ₄ + 4 g Ag ₂ SO ₄	47.5
(7) 95.5 percent H ₂ SO ₄ + 10 g P ₂ O ₅	54
(8) H ₂ SO ₄ + P ₂ O ₅ + Ag ₂ SO ₄	62
(9) 98 percent H ₂ SO ₄ + 0.5 g Ag ₂ SO ₄	84
(10) 98 percent H ₂ SO ₄ + 1 g Ag ₂ SO ₄	98.5
(11) 98 percent H ₂ SO ₄ + 2 g Ag ₂ SO ₄	100

(3) 30 g of P₂O₅ added to 100 cc of concentrated H₂SO₄.

(4-7) Amount of inorganic catalyst added to each 100 cc.

(9-11) Amount of inorganic catalyst added to each 100 cc.

(8) Similar to 7, but 2 g Ag₂SO₄ added to each 100 cc.

24 Towne, C. C., Work cited (see footnote 14).

25 Kattwinkel, R., Work cited (see footnote 10).

A number of experiments were carried out to ascertain the catalytic effect²⁶ of various inorganic substances in concentrated sulphuric acid. Several possible sulphonation catalysts (mercuric sulphate, sodium sulphate, lead sulphate, ferrous sulphate, nickelous sulphate hexahydrate, stannic sulphate, uranium nitrate, thorium nitrate, uranium plumbate, tungstic acid, cuprous oxide and tantalic acid) were studied and found not to have any appreciable effect under the conditions employed. When the proposed catalyst was sufficiently soluble, 2 grams was dissolved in 100 cc of concentrated acid, and 60 cc of the resulting solution was shaken for 5 minutes in the usual manner with 20 cc of benzene. In most instances the inorganic salts were sparingly soluble, and a concentrated solution of the salt in 95.5 percent sulphuric acid was employed.

To find whether the beneficial effect of silver sulphate in sulphonating reagents is sufficient to warrant its use in analytical procedures, the experiments leading to the data in table 5 were carried out. The coal-tar distillate used in this study was treated with 80 percent acid and distilled to its original endpoint to remove olefins and polymers before being sulphonated. The oil was then shaken in the usual manner with 3 volumes of acid (60 cc), and later with two 15-cc portions. In the two latter sulphonations the rise in temperature was slight. Kattwinkel's reagent also was used for the purpose of comparison. The results (table 5) indicate that the concentration of acid is most important and that the presence of silver sulphate increases the reactivity of the sulphonating reagent very little. A second washing with the silver-free reagents gave results almost identical with those obtained by one treatment with the silver sulphate-sulphuric acid solutions.

Reactions of Sulphonating Reagents With Saturated Hydrocarbons

There is considerable disagreement concerning the stability of saturated hydrocarbons toward sulphonating reagents, but it is recognized that straight-chain paraffins are most resistant. The behavior of several representative saturated hydrocarbons in the presence of the sulphonation reagents described above was investigated.

As in previous experiments, 1 volume (20.0 cc) of the hydrocarbon was shaken vigorously for 5 minutes with 3 volumes of the acid reagent at room temperature. In most cases there was at least a slight reaction, which was indicated by loss in volume of the hydrocarbon, discoloration, and elevation of temperature. In general, the branched-chain hydrocarbons were most reactive (see table 6).

The results show that the Kattwinkel solution and 100-percent sulphuric acid attack saturates too readily to be suitable as analytical reagents. It has been claimed²⁷ that saturated hydrocarbons are appreciably soluble in sulphonating mixtures, but in view of the fact that the large volume losses observed in the present work were accompanied by the generation of considerable heat (table 6) it appears that chemical action is mainly responsible for loss of oil²⁸. If extraordinarily reactive hydrocarbons, such as isooctane, are excluded, the other reagents (95.5, 98 percent acid and 98 percent acid with silver sulphate) apparently give results as accurate as those obtained in the other stages of hydrocarbon oil analysis.

26 Manning, A. B., and Shepherd, F. M. E., The Determination of Aromatic, Unsaturated, and Naphthene Hydrocarbons in Light Oils and Motor Spirits: Fuel Research Tech. Paper 28, 1930, 14pp. Ambler, J. A., and Cotton, W. J., The Use of Catalysts in the Sulphonation of Aromatic Compounds: Ind and Eng. Chem., vol. 12, 1920, pp. 968-969.

27 Thole, F. B., Estimation of Benzene and Toluene in Petroleum: Jour. Soc. Chem. Ind., vol. 38, 1919, pp. 39-43T.

28 A careful study by other workers (Morgan, J. J., and Soule, R. P. The Examination of Low-Temperature Coal Tars. I.: Ind. Eng. Chem., vol. 15, 1923, pp. 587-591) has shown that the solubility of saturated hydrocarbons in sulphonio acids is negligible.

TABLE 5.- Sulphonation of tar oil: Effect of silver sulphate

Acid reagent	Amount of acid, cc	Undissolved oil, cc	Total sulphonation, percent
(1) 96 percent H_2SO_4	60	3.5	82.5
	15	3.35	83.25
	15	3.3	83.5
(2) 96 percent $H_2SO_4 + Ag_2SO_4$	60	3.4	83.0
	15	3.25	83.75
	15	3.15	84.25
(3) 98 percent H_2SO_4	60	3.2	84.0
	15	3.1	84.5
	15	3.0	85.0
(4) 98 percent $H_2SO_4 + Ag_2SO_4$	60	3.1	84.5
	15	3.0	85.0
	15	2.9	85.5
(5) Kattwinkel's reagent	60	3.1	84.5

(2,4) 2 g Ag_2SO_4 added to each 100 cc.

(5) 30 g P_2O_5 added to 100 cc concentrated H_2SO_4 .

When 60 cc of acid reagent was used the temperature rose to 39° to 43°.

TABLE 6.- Acid reagents on saturated hydrocarbons

Hydrocarbon	95.5%	98%	98% H_2SO_4	100% H_2SO_4		$H_2SO_4 + P_2O_5$	
	H_2SO_4 ,	H_2SO_4 ,	+ Ag_2SO_4 ,			(Kattwinkel)	
	percent	percent	percent	Temperature	percent	Temperature	percent
	loss	loss	loss	rise, °C.	loss	rise, °C.	loss
(1) n-Heptane.....	0.3	0.3	1.0	0.6
(2) Cyclohexane.....	.5	1.1	1.5	2.35
(3) Methylcyclohexane.....	.65	.6	1.0	5.0	10.6	5	4.8
(4) Decalin.....	.2	.6	.6	8.0	8.5	10	7.4
(5) p-Menthane.....	.1	.25	.2	3.5	5.85	3.5	2.2
(6) Isooctane.....	1.5	2.85	5.0	12.65
(7) Diisoamyl.....	.45	.75	.6	2.0	1.25

The H_2SO_4 - Ag_2SO_4 was prepared by adding 2 g Ag_2SO_4 to 100 cc of acid. Kattwinkel's reagent contained 30 g P_2O_5 and 100 cc 95.5 percent H_2SO_4 .

(4) Decahydronaphthalene.

(5) 1-Methyl-4-isopropylcyclohexane.

(6) 2,2,4-Trimethylpentane.

(7) 2,7-Dimethyloctane.

Information on the resistance of saturated hydrocarbons to sulphonation in the presence of aryl sulphonic acids was obtained by treating synthetic solutions of saturates and aromatics with 98 percent sulphuric acid and with acid of the same strength containing silver sulphate (2 g per 100 cc of acid). The hydrocarbon solutions thus studied varied in aromatic content (benzene or toluene) from 50 to 90 percent and contained decalin, isooctane, p-menthane, methylcyclohexane, or diisoamyl as the saturated component. The experimental procedure employed was similar to that used with the hydrocarbons of table 6. The results obtained are in agreement with the conclusions drawn previously from the data in table 6. Considerable amounts of isooctane and methylcyclohexane were removed along with the aromatics. Because of the inertness of the saturated components, the separation of aromatics and saturates was more satisfactory for the solutions containing decalin and diisoamyl. The data obtained also indicate that the use of silver sulphate is not decidedly advantageous and that a second washing with the acid reagent is necessary to remove all the aromatics. A double extraction procedure for removing the last traces of aromatics might be advantageous, if conducted properly, in that smaller amounts of saturates would be removed. For example, 3 volumes of 96 percent acid in the preliminary step would extract nearly all the aromatics (see table 5) and attack smaller quantities of saturates than a similar treatment with 98 percent acid. The second washing with a smaller quantity of 98 percent acid (3 volumes based on residual oil instead of original oil) should remove only negligible amounts of saturates, since the amount of acid is less and the elevation of temperature in the second washing is small.

COMPARISON OF TYPICAL SULPHURIC ACID METHODS

Two synthetic mixtures of known composition were analyzed by several sulphuric acid methods previously recommended and by four modifications of the Kester and Pohle method²⁹ that were expected to give results of greater accuracy. These data are presented in table 8.

In order that the synthetic solutions employed in these analyses might resemble the complex neutral oils obtained from coal, several different representatives of each hydrocarbon group were included, and the aromatic contents were made high. Relatively reactive olefins were employed in solution 1, while solution 2 (see table 7) contained a considerable quantity of an unreactive olefin (diamylene) as well as unsaturated hydrocarbons more easily attacked by sulphuric acid.

The high olefin values obtained by the Morrell and Levine method³⁰ (olefins determined with 91 percent acid and distillation) indicate that the distillation residue was too great for substitution in their empirical formulas. The large distillation residue probably was due to the presence of high-boiling alkylated aromatics. Petroleum distillates containing small amounts of aromatics might yield only negligible amounts of such condensation products when treated with 91 percent acid, but the formation of considerable amounts of alkylated aromatics is to be expected in aromatic-rich synthetic solutions and coal-tar oils.

Towne³¹ extracted olefins with 93 percent acid that had been cooled previously to 35° F. Another sample of oil then was treated with 98 percent acid and distilled to the original endpoint. From the data obtained, the olefin and aromatic contents were calculated.

29 Kester, E. E., and Pohle, W. D., Determination of Olefin, Aromatic, and Paraffin Hydrocarbons in Neutral Oil from Coal Tar: Ind. Eng. Chem., Anal. Ed., vol. 3, 1931, pp. 294-297.

30 Morrell, J. C., and Levine, I. M., Approximate Determination of Olefins and Aromatic Hydrocarbons: Ind. Eng. Chem., Anal. Ed., vol. 4, July 1932, pp. 321-322.

31 Towne, C. C., Work cited (see footnote 14).

The results in table 8 indicate that aromatics were removed along with the olefins in the treatment with 93 percent sulphuric acid. This is not astonishing in view of the high aromatic contents of the synthetic solutions and the tendency of 93 percent acid to sulphonate aromatic hydrocarbons.

TABLE 7.- Composition of synthetic solutions

Constituent	Amounts of constituents, cc	
	Solution 1	Solution 2
Olefin mixture ¹	300	300
Diamylene.....		200
p-Cymene.....	25	25
Ethylbenzene.....	25	25
Diethylbenzene.....	25	25
Benzene.....	625	625
Decahydronaphthalene.....	150	100
Heptane.....	150	100

Solution 1: 15% olefins, 70% aromatics, 15% saturates.

Solution 2: 25% olefins, 65% aromatics, 10% saturates.

¹9 cc diisobutylene, 26.5 cc styrene, 33.5 cc octylene, 156 cc cyclohexene, 178 cc pinene, and 238 cc dipentene.

The results obtained with the Kester and Pohle method³² indicate that their olefin reagent (80 percent acid) is satisfactory for neutral oils of the type derived from coal if only small quantities of unreactive olefins are present. With solutions containing appreciable amounts of unreactive olefins, it is probable that low olefin and high aromatic values would result. Since there is no evidence of the presence of large quantities of unreactive olefins in coal-tar oils (known to have high aromatic contents), the Kester and Pohle analysis is to be preferred to the other methods of this type that employ concentrations of sulphuric acid much over 80 percent as olefin reagents.

The Ipatieff and Pines method³³, employing concentrated acid at 0° C. as the olefin reagent, gave such a high value for the olefin content that the analysis was not continued. Because of the heat liberated when synthetic solution 1 was treated with concentrated acid, the temperature at times was higher than 0° C. Although a lower olefin value might have been obtained in this experiment by more efficient cooling, concentrated sulphuric acid attacks aromatics too rapidly to be used for oils containing high concentrations of aromatic hydrocarbons.

The other procedures used to analyze the two synthetic solutions were modifications of the Kester and Pohle method. Using 82 percent instead of 80 percent acid (modification A) gave results that were more satisfactory for solution 2 but less for solution 1. Because of data presented earlier (see table 3), it was believed that using two reagents (80 and 82 percent acid) would be advantageous. Accordingly, 80 percent acid was used for reactive olefins (modification B), and the residual oil was treated with 82 percent acid to remove more unreactive olefins. This scheme gave values almost identical with those obtained by modification A.

32 Kester, E. B., and Pohle, W. D., Work cited (see footnote 29).

33 Ipatieff, V. N., and Pines, H., Polymerization of Ethylene Under High Pressures in the Presence of Phosphoric Acid: Ind. Eng. Chem., vol. 27, Nov. 1935, pp. 1364-1369.

TABLE 8.- Comparison of several sulphuric acid methods (volume percent)

Method of analysis	Solution 1			Solution 2		
	Olefins	Aromatics	Saturates	Olefins	Aromatics	Saturates
Constituents present.....	15	70	15	25	65	10
Morrell and Levine.....	23.4	66.6	10.0	36.8	56.2	7.0
	24.3	65.2	10.5	36.0	56.5	7.5
Towne.....	30	55	15	30	59.5	10.5
	29	57	14	31	59	10.0
Kester and Pohle.....	16	69.3	14.7	18	68.8	13.2
	17	68.6	14.4	18	70.4	11.6
Ipatieff and Pines.....	48					
Modification A.....	20	67.5	12.5	22	66.4	11.6
Modification B.....	20.5	67.3	12.2	21.5	63.0	15.5
Modification C.....	21	66	13	20.5	65.9	13.6
				20	67.1	12.9
Modification D.....				23	63.3	13.7
					¹ 66.8	¹ 10.2

Modifications of Kester and Pohle method:

- A.- Olefins determined with 82 percent instead of 80 percent H_2SO_4 .
- B.- Olefins determined with 80 percent followed by 82 percent H_2SO_4 .
- C.- Olefins determined with 80 percent H_2SO_4 in presence of cresol.
- D.- Olefins determined with 80 percent followed by 82 percent H_2SO_4 in presence of cresol

¹Aromatics removed with 98 percent H_2SO_4 containing Ag_2SO_4 .

Cresol was added to the oil in modification D with the expectation that condensation with the unsaturated hydrocarbons would cause increased amounts of olefins to be removed as high-boiling alkylated cresols. Higher olefin values were actually obtained in the presence of cresol, but this method apparently is not superior to the other modifications described in table 8. Modification D is identical with B, with the exception that cresol was added before the oil was treated successively with 80 and 82 percent acids. This modification gave the best analysis of all for solution 2, which contained unreactive as well as re-active olefins.

Although all the modifications of the usual sulphuric acid analysis considered in table 8 failed to give completely satisfactory analyses, a new method of the graphic type was found to indicate accurately the olefin and aromatic contents of both solutions. The graphic method is based on the changes in volume and physical constants that accompany successive extractions of the oil with sulphuric acid solutions of increasing concentration. An account of this work³⁴ will be published elsewhere.

34 Fisher, C. H., and Eisner, Abner, Graphic Analysis of Hydrocarbon Oils. The Volume-Physical Constants Relationship Resulting from Successive Extractions with Sulphuric Acid: Ind. Eng. Chem., Anal. Ed., vol. 9, August 1937, pp. 366-370.

SUMMARY AND CONCLUSIONS

In the determination of olefins, reagents such as 80 percent sulphuric acid, while quite effective with reactive olefins, fail to dissolve or polymerize the more unreactive olefins (diamylene, cetene, etc.). Sulphuric acid of higher concentration is more effective but its use is limited because of its tendency to attack aromatics and catalyze the interaction of olefins and aromatics. Acid of concentration slightly higher than 80 percent may be used satisfactorily for oils of low aromatic content. Coal-tar oils containing large quantities of aromatic hydrocarbons require weak olefin reagents, such as 80 to 82 percent acid. When 80 percent acid is used with oils containing unreactive ethylenic hydrocarbons, low olefin and high aromatic values may be expected.

Previous claims that the addition of silver sulphate or boric acid to sulphuric acid gives olefin reagents of enhanced effectiveness have been confirmed. However, these inorganic substances cannot be used to advantage because other reactions of an undesirable nature are promoted. For example, silver sulphate catalyzes both the sulphonation of aromatics and the condensation of olefins with aromatics. Although 80 percent acid attacks only olefins, the addition of silver sulphate gives a reagent that causes olefins to react with aromatics. A moderately dilute sulphuric acid solution containing both silver sulphate and boric acid was found to be quite effective in dissolving and polymerizing olefins but not capable of sulphonating aromatics. However, this solution cannot be used to determine olefins, since it causes olefins to condense with aromatic hydrocarbons.

Sulphuric acid of concentrations up to about 84 percent may be employed with only negligible sulphonation of aromatics. However, acid of concentration as low as 82 percent was observed to cause the alkylation of aromatics. This demonstrates again that the greatest objection to using stronger olefin reagents is the interaction of olefins and aromatics.

Kattwinkel's olefin reagent (concentrated sulphuric acid containing boric acid) attacks olefins vigorously, but as it also sulphonates aromatics inaccurate results may attend its use.

Concentrated sulphuric acid was found to be quite effective in sulphonating 15 aromatic hydrocarbons. Benzene and certain benzene derivatives possessing large and bulky alkyl substituents offered most resistance in sulphonation. A sulphuric acid solution of higher concentration (98 percent) was observed to sulphonate aromatics with ease. Concentrations higher than 98 percent and Kattwinkel's aromatic reagent (96 percent acid containing phosphorous pentoxide) should not be used as reagents, since such solutions remove considerable amounts of saturated hydrocarbons. Silver sulphate may be added to increase the sulphonating action of aromatic reagents, but its use is not necessary or very advantageous. One washing with three volumes of either 96 or 98 percent acid was found to remove nearly all the aromatics from coal-tar oils. Two washings with about 3 volumes of 98 percent acid (or one washing with 96 percent acid followed by treatment with 98 percent acid) are recommended, since this procedure removes all aromatics without attacking more than traces of saturated hydrocarbons.

Using two synthetic solutions, several previously described sulphuric acid methods for determining olefins and aromatics were compared. Although no one method was completely satisfactory, the Kester and Pohle procedure (olefins determined with 80 percent acid) gave the most accurate results.

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DECEMBER 1937

UNITED STATES
DEPARTMENT OF THE INTERIOR
HAROLD L. ICKES, SECRETARY

BUREAU OF MINES
JOHN W. FINCH, DIRECTOR

REPORT OF INVESTIGATIONS

PROGRESS REPORTS - METALLURGICAL DIVISION

20. ANNUAL REPORT OF THE METALLURGICAL DIVISION,
FISCAL YEAR 1936-37

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BY

R. S. DEAN AND OTHERS

December 1937

REPORT OF INVESTIGATIONSDEPARTMENT OF THE INTERIOR - BUREAU OF MINES

PROGRESS REPORTS - METALLURGICAL DIVISION

20. Annual Report of the Metallurgical Division, Fiscal Year
1936-37.¹By R. S. Dean² and others

FOREWORD

This report, the third of a series of annual reports of the Metallurgical Division to be released in this form, covers the fiscal year July 1, 1936, to June 30, 1937.

The technological work of the division is performed by sections. Thus, an efficient organization is provided to prosecute the various problems under investigation, and a logical system is set up for presenting the material contained in this report.

Several outstanding achievements have resulted from the investigations of the Metallurgical Division during the past year:

The most spectacular development was the application of high-frequency, standing sound waves to the flocculation and removal of dust, smoke, and fume from gaseous suspensions. This has aroused the interest of a wide variety of industries and communities in this country and abroad that are bent on the recovery of valuable products that now go up the stack or are faced with the problem of preventing air pollution. This method appears to have possibilities for the solution of these problems and only awaits a practical method for the production of suitable standing waves.

Another interesting feature, the development of a new method for effecting the separation of gases of different densities or molecular weight by means of diffusion methods, may have a definite and practical significance. It provides a mechanical means for separating gases in a manner that is as simple fundamentally as the conventional separation of mineral particles by gravitational methods.

Recent findings regarding a novel process for the treatment of chromite ores by chlorine followed by the reduction of the chromium chlorides with hydrogen indicate that pure chromium metal may be produced cheaply.

¹/ The Bureau of Mines will welcome reprinting of this paper, provided the following footnote acknowledgment is used: "Reprinted from U. S. Bureau of Mines Report of Investigations 3357."

²/ Chief engineer, Metallurgical Division, U. S. Bureau of Mines.

The results of the work of the Division show that its function of conducting investigations relating to the treatment of mineral products from the state in which they are mined to the refined product has been prosecuted assiduously. The logical procedure, starting with highly theoretical implications through laboratory development of new methods and concluding with plant testing to the point of demonstrating practical application, has been followed. This provides incentive to the establishment of new industries. Examples of recent new operations based on results obtained by the division are the installation of a flotation unit to treat scheelite slimes and the construction of a mill to separate talc and magnesite by flotation.

The U. S. Bureau of Mines has received many requests for copies of recent papers dealing with metallurgical investigations. Excerpts from the Bureau's reports on manganese and fume dispersion were printed in scientific journals in several foreign countries.

Because of the frequent demand for such information and because most of the Bureau of Mines publications on the problems are out of print or out of date, the Metallurgical Division has prepared the following Information Circulars, which may be available this year: "Occurrence and Treatment of Mercury Ores at Small Mines", "Smelting Ores in the Electric Furnace", "The Treatment and Sale of Black Sands", and "Laboratories that Make Assays, Analyses, and Tests on Ores, Minerals, Metals, and Other Substances."

New and important additions to the equipment of the Metallurgical Division were electric furnaces and accessories of various types and complete equipment for concentrating ores on a large scale in the Boulder City plant of the Electrometallurgical Section and a universal microscope with 150 special interchangeable attachments for different studies of ores and mill products in the Salt Lake City laboratory of the Nonferrous Metallurgical Section.

The special studies section of the Metallurgical Division now has office space and laboratory equipment in the new Eastern Experiment Station of the Bureau of Mines at College Park, Md.

ACKNOWLEDGMENT

This report has been prepared from statements submitted by the following members of the Metallurgical Division staff:

- C. W. Davis, assistant to the chief engineer and supervising engineer, ore-testing section.
- C. G. Maier, supervising engineer, metallurgical fundamentals section.
- V. H. Gottschalk, supervising engineer, special studies section.

- J. Koster, supervising engineer, electrometallurgical section.
- W. H. Coghill, supervising engineer, ore-dressing section.
- G. L. Oldright, supervising engineer, non-ferrous metallurgy section.
- E. S. Leaver, supervising engineer, precious metals section.
- F. S. Wartmen, acting supervising engineer, metallurgy of copper section.
- C. E. Wood, acting supervising engineer, blast-furnace studies section.
- B. A. Rogers, supervising engineer, metallurgy of steel section.

Acknowledgment is made of assistance rendered by the following cooperating institutions:

- University of Alabama, Tuscaloosa, Ala.
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- University of Maryland, College Park, Md.
- University of Minnesota, Minneapolis, Minn.
- University of Missouri, Rolla, Mo.
- University of Nevada, Reno, Nev.
- University of Utah, Salt Lake City, Utah.

R. S. DEAN,
Chief Engineer,
Metallurgical Division.

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METALLURGICAL FUNDAMENTALS SECTION

The work of the metallurgical fundamentals section during the past year, representing active experimental investigation or thermodynamic study, may be summarized as the following problem subjects:

1. Low-temperature specific-heat investigations.
2. Development of a method of concentrating gases.
3. Thermodynamic properties of metallic carbides and nitrides.
4. Production of pure chromium from chromium chlorides.
5. Thermodynamic properties of the hydrates of calcium sulphate.

While this list is not identical with officially designated problems, due to overlapping of subjects and to new developments within specified problems, presentation upon the basis of this subdivision will best serve to review the work of the section.

The first of these items represents a continuing investigation of the program of determining low-temperature specific heats of materials of special metallurgical interest or directly pertinent to theoretical problems of metallurgical importance.

The second item is the outgrowth of a problem reported as new last year, proposing the study of methods of producing pure hydrogen from hydrocarbon gases, and the success achieved in solving a difficulty in the original problem has so overshadowed the latter in general importance as to dominate the experimental work in connection with it.

The study of the thermodynamic properties of metallic carbides and nitrides is the eighth and the last contemplated member of the series of "Contributions to the Data on Theoretical Metallurgy", which have been prepared by the section, each one describing the application of thermodynamic methods to a special field.

Previous work of the section had resulted in substantially complete thermodynamic description of the properties of anhydrous chromium chlorides and the development of methods, based on these findings, of continuously chlorinating chromite ores in technically feasible units with the production of easily purifiable and virtually anhydrous chromic chloride. The fourth item of the list represents an investigation of possible means of utilizing this material for direct reduction to pure chromium, the theoretical possibility of doing so in a technically advantageous way having been disclosed by the thermodynamic studies first mentioned.

The fifth and last item represents the overlapping of several fields of investigation, including low-temperature specific-heat investigations and thermodynamic interpretation of equilibria. Local representatives of the gypsum plaster industry, which has shown steady and rapid growth in the last two decades, frequently have consulted the staff of the section concerning various physical chemical properties of calcium sulphate hydrates; and the realization that thermodynamic information in this case would be pertinent to a number of practical problems in production and use justified a preliminary survey whose later developments will be discussed below.

At the end of this discussion of the work of the section will be found a list of publications in which not only current projects are included but also reporting the status of those previously prepared manuscripts whose final disposition had not been disclosed by last year's report.

1. Low-Temperature Specific-Heat Investigations

Materials investigated by low-temperature specific-heat measurements during the year were selenium glass and selenium crystals, tellurium, and calcium sulphate hemihydrate in one of its forms. None of these materials had been measured previously. While the technical use of selenium has increased surprisingly in recent years, the immediate interest in selenium and tellurium was scientific rather than directly technical. Previous measurements by the section of the specific heats of manganese oxide and sulphide had shown both these materials to have specific-heat anomalies at low temperatures, and from this it was postulated that similar anomalies might exist in manganese selenide and telluride. The exact nature of such anomalies is not understood at present, although certain but not all of them are associated with magnetic changes. They are, in fact, more common than would first seem probable and, perhaps, especially so in the case of the compounds of those metals that are often constituents of alloy steels.

The importance of these anomalies from a thermodynamic viewpoint resides in the fact that it is not clear, so far, what effect the anomalous specific heat and its corresponding entropy may have on the calculation of reduction or other equilibria of metallurgical importance which it is desired to calculate through the medium of entropies derived from low-temperature specific-heat measurements. It is thought that by studying the variations produced in the anomaly by substituting various members of the sixth periodic group of metalloidal elements in the manganese group, some clue might be furnished as to the true significance of the anomalous specific heat and the method of dealing with it in thermodynamic calculations.

The measurements upon calcium sulphate hemihydrate were part of a series on different forms of hydrous calcium sulphates, selenite and natural anhydrite having been reported in last year's record. These served for a preliminary review, which will be discussed under the last item of the section list.

A number of papers presenting results obtained in previous work but not yet published were prepared also, and include measurements upon chromium and chromium oxide, chromous and chromic chloride, molybdenite, and pyrite.

New samples now ready for experimental work are manganese in specially purified powder form, the alpha form of calcium sulphate hemihydrate, and soluble anhydrite, while manganese selenide and telluride are still in course of preparation.

2. Development of a Method of Concentrating Gases

In last year's report, attempts to improve the thermal decomposition of methane for the production of hydrogen gas by catalytic means were recorded. Although it had, indeed, been found possible to accomplish a part of the original purpose of the problem, that of controlling the form of carbon produced and although certain materials were found to exercise a definite catalytic influence upon the thermal decomposition, the results did not show marked improvement over existing methods, especially as concerns obtaining completeness of conversion. At this point the question was taken up whether means of concentrating gases by mechanical methods could be brought to a stage of technical utility.

A start was made with an attempt to modify a molecular diffusion pump using water vapor as energizing medium, and definite evidence was found that such pumps were capable of demonstrating a real concentration effect in gases even when operated at near atmospheric pressures, although they are truly effective in this sense only when operated at relatively low pressures below 1 or 2 millimeters of mercury.

An intensive study of the behavior of such a pump from the basis of kinetic gas theory soon elucidated the reasons for the low efficiency of a conventional type of diffusion pump when operated at atmospheric pressure and gave definite hints as to the means by which its deficiencies might be overcome under these conditions. After a number of preliminary types of apparatus modified in the indicated way were tested and their mechanical faults eliminated one by one, a unit was built that operated with an efficiency of 85 to 95 percent of the theoretical maximum possible concentration allowed by kinetic gas theory for the particular materials being separated.

What has been accomplished is, in essence, the provision of mechanical means for separating gases of different densities or molecular weights in a manner no more complex than the conventional separation of mineral particles by gravitational methods. The unit finally evolved may be considered either as a diffusion pump so modified as to be effective at barometric pressures, or as a filter having passages of the size of intermolecular spaces in gases under these conditions of pressure, but also arranged so as to be incapable of plugging or loss of permeability. While the process of diffusion under normal circumstances always tends to produce uniformity of composition of fluid mixtures, there has been developed, and it is believed for the first time, a means of utilizing the different rates of diffusion of gases of

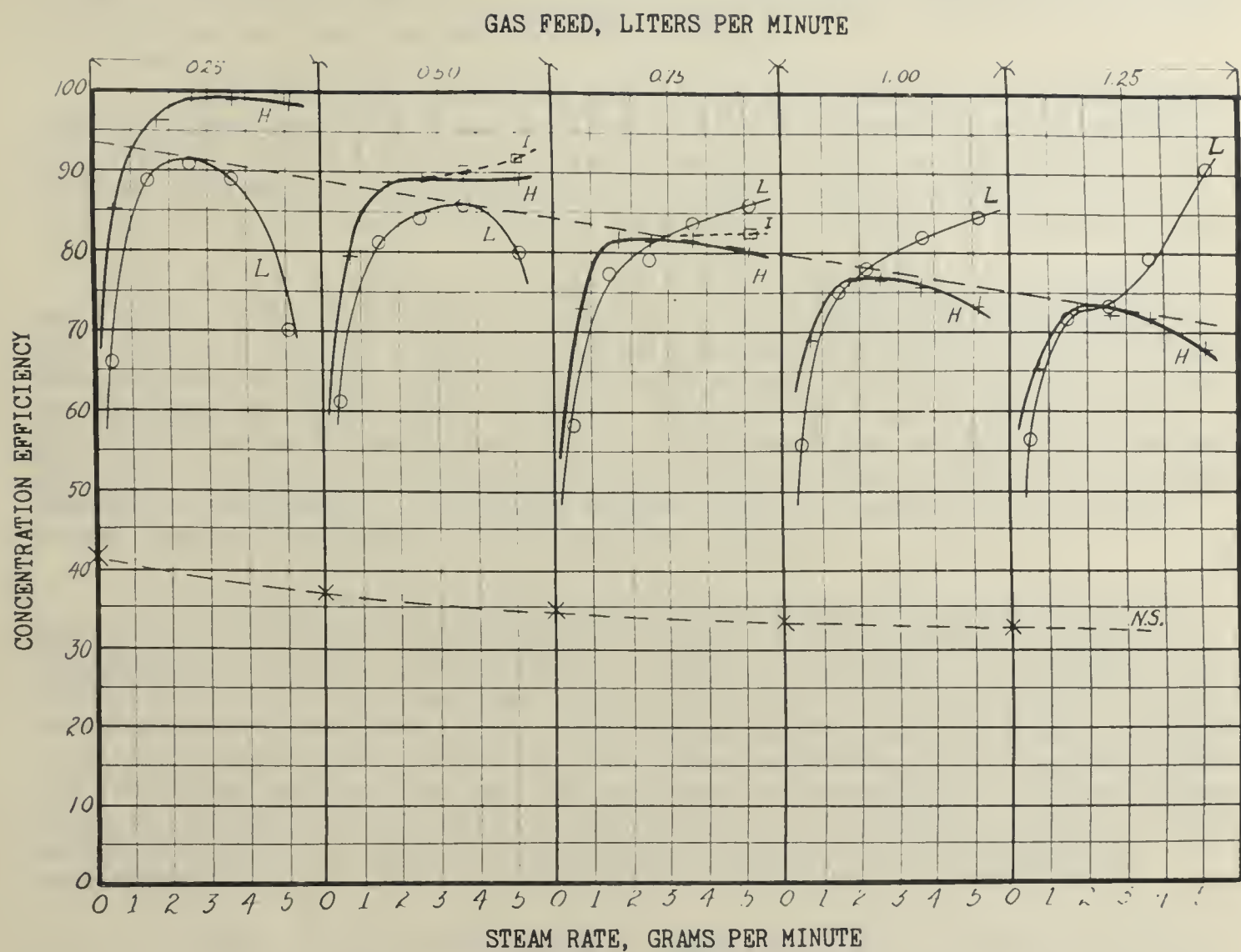


Figure 1.- Characteristic curves of gas concentrator.

These curves show the concentration efficiency at various steam rates for a number of particular gas feed rates of the test unit. The H curve is for high throttling factors, the L for low, and the I branch for intermediate. The dotted line NS represents the slight concentration obtained when no steam is used.

different densities in such a way as to bring about a separation of constituents rather than uniformity of composition. While the phenomena and physical relationships of gas-diffusion rates have been well-known and their interpretation considered one of the major triumphs of the classical kinetic gas theory, an efficient means of utilizing the principles involved for technical purposes has not been demonstrated heretofore.

More than 400 individual tests have been made on the small experimental demonstration unit, and while it cannot be claimed that the method does not have sharply definable limitations, the tests have continued to confirm the efficacy of the method and its ultimate convertibility into a new tool for the metallurgist or chemist dealing with gaseous or vaporous media.

So far the tests have been confined to the separation of hydrogen gas from mixtures of hydrogen and nitrogen corresponding to a manufactured gas. Previous work of the metallurgical fundamentals section in connection with the production of sponge iron by natural-gas reduction had resulted in the development of extremely efficient and economical means of converting mixtures of natural gas and air into a "re-formed" gas containing approximately 30 percent hydrogen, the remainder being mostly carbon monoxide and nitrogen. Because of similarity of molecular weights, carbon monoxide and nitrogen behave nearly identically in diffusion processes.

Typical results of the concentration process on a gas containing approximately 30 percent hydrogen in nitrogen are illustrated in figure 1. The curves are shown in five sections, each corresponding to a specified rate of gas fed to the unit, and show the variation of the concentration efficiency as a function of the steam rate. The curves marked H are for high throttling factors and those marked L are for low. The throttling factor is defined as the percentage of the total gas fed separated as concentrate, and the concentration efficiency is the ratio of the grade of gas actually separated to the maximum possible according to kinetic gas theory for the corresponding throttling factor.

Such a concentration process results in a 50- to 60-percent H_2 gas concentrate in the first step starting from 25 to 30 percent, and this concentrate may be taken to 85 to 95 percent in a second step. From the practical aspect, the advantages of this method of preparing hydrogen are in the matter of low first cost of the apparatus and simplicity of operation and control, which result in the indicated possibility of making 95 percent hydrogen at a cost not far from three times that of natural gas, which in some localities may be no more than 50 cents per 1,000 cubic feet of hydrogen produced.

At present, a series of "characteristic" curves is being obtained suitable to determine the various factors entering into the engineering design of gas concentrators for particular purposes. Its application to types of separation other than the enrichment of hydrogen has not yet been attempted, but a number of interesting possibilities are obvious.

3. Thermodynamic Properties of Metallic Carbides and Nitrides

Since a manuscript presenting correlated thermodynamic data for sulphur, its oxides, metallic sulphides and sulphates had been completed near the beginning of the year, it became apparent that one metallurgically important group of compounds that had not previously been critically reviewed comprised metallic carbides and nitrides. The thermodynamic properties of such materials are involved not only in connection with questions as to their stability and form in alloys but are also especially important in connection with the reduction metallurgy of a number of alloying metals, perhaps no more so than in the case of chromium. During the year a critical recalculation of existing data on the thermodynamics of this group of compounds was completed and a manuscript was prepared presenting not only correlated data of this type for individual compounds but also extending the application of the figures to a number of technical or practical examples. Thus, it was possible to show direct application of the data to such technical problems as the use of calcium carbide for debismuthizing lead bullion, gas case-hardening of ferrous materials, the direct synthesis of ammonia from methane and air, the behavior of manganese in mild steels, and the synthetic production of cyanide from ammonia and carbon monoxide.

4. Production of Pure Chromium from Chromium Chlorides

Mention has been made in the introductory paragraphs that the thermodynamic theory of the reduction of chromium chlorides had been worked out previously and means developed for technically producing such chlorides directly and continuously from chromite ores. The possibilities inherent in reduction by such means will be clear from a comparison of approximately 3 percent reduction per cycle when hydrogen gas is passed over chromous or chromic chloride at 800°C ., as against a few tenths percent possible when the same gas is passed over chromium oxide at temperatures of $1,400^{\circ}$ to $1,500^{\circ}\text{C}$. Thus, gas reduction, which seems logically suited to produce carbon-free chromium, must be regarded as technically unpromising, even if physically and chemically possible when starting from oxide, both from the point of view of construction of suitable apparatus and excessively low chemical efficiency, but definitely within the bounds of technical feasibility when the anhydrous chloride is the starting point.

In view of the theoretical background already attained, the specific problem involved becomes almost entirely a study of the feasibility of setting up a circulating hydrogen system in which hydrogen of high purity may be maintained in continuous countercurrent contact over chromium chloride, and with provision for removing the product of the reduction, hydrochloric acid gas, without introducing such impurities as water vapor, oxygen, nitrogen, carbon dioxide, or hydrocarbons. Since the gases produced by the reduction of chromium do not build up enough concentration to exceed the reduction equilibria of iron chlorides, the question of materials of construction is not a major problem and certain types of steel are indicated as entirely suitable materials of construction.

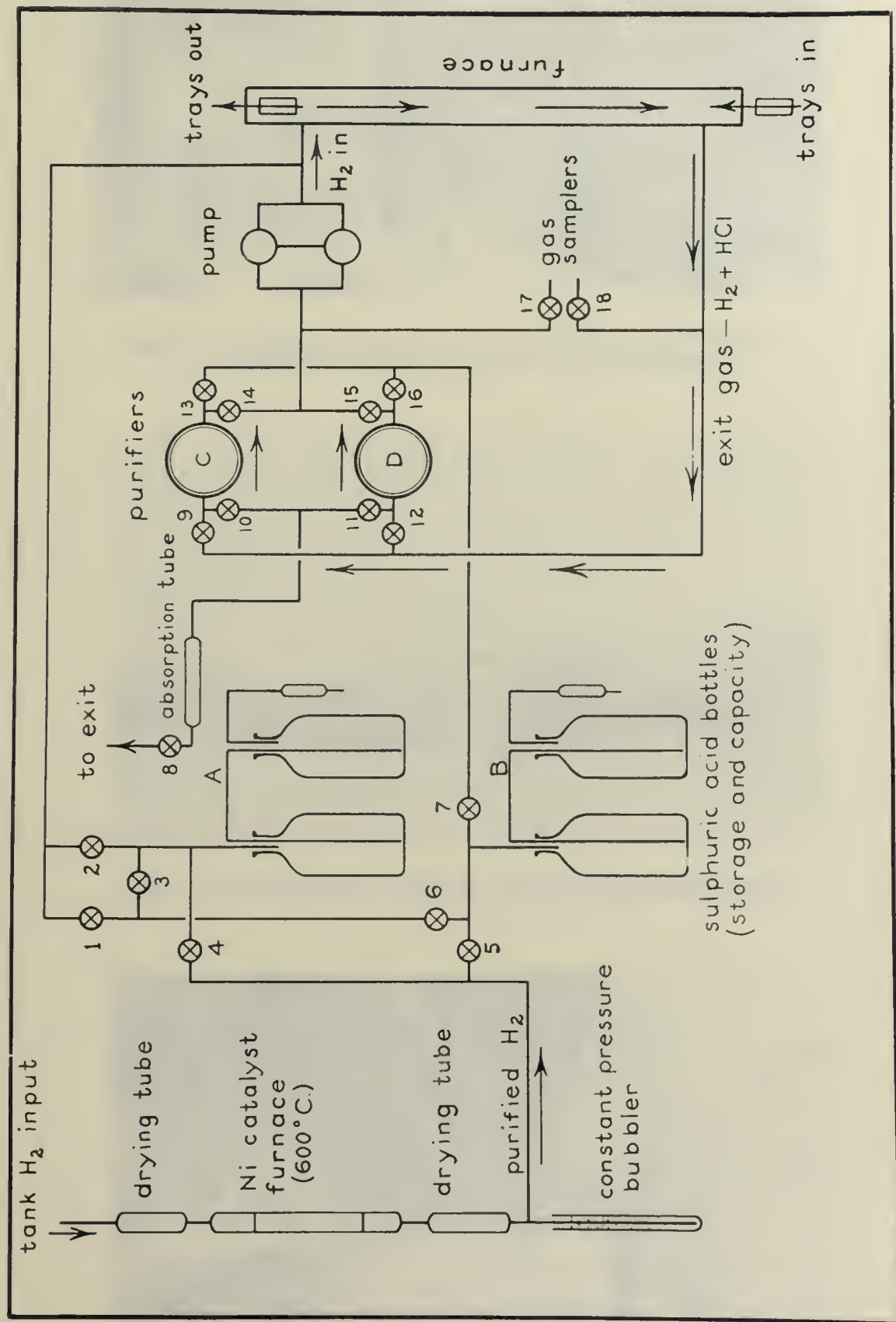


Figure 2.—Diagrammatic flow sheet of continuous unit for reducing chromic chloride by hydrogen.

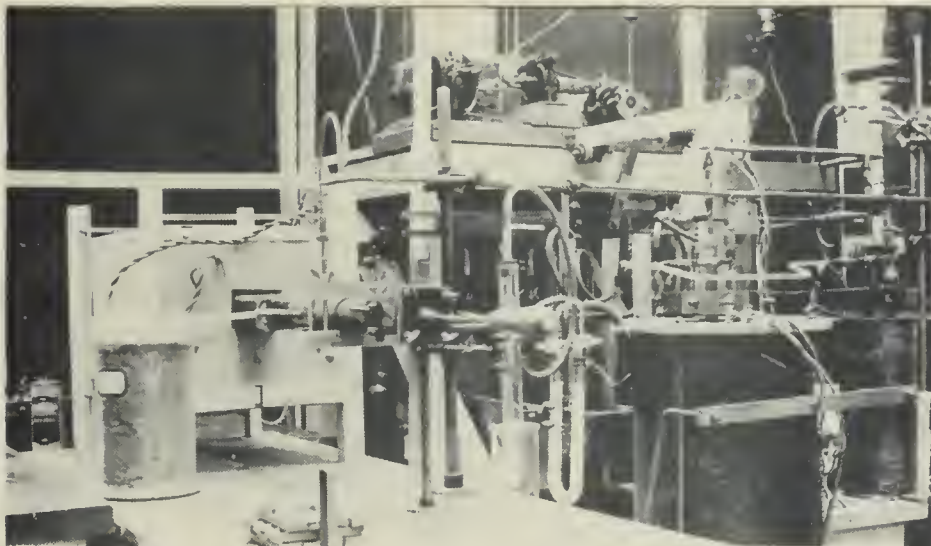


Figure 3.—Photograph of apparatus shown in flow sheet (fig 2). Purifiers at right, reduction furnace at left, and pump in background.



Figure 4.—The building as acquired for electrometallurgical investigations.



Figure 5.—Interior of the building as acquired.

Preliminary experiments showed that the cyclical absorption and desorption of hydrochloric acid gas on active carbon between room temperature and 200° to 300°C. served to remove the hydrochloric acid from the hydrogen, and that the carbon would absorb approximately 4 percent its weight of HCl gas per cycle before requiring desorption.

After a series of cyclical tests that seemed to confirm the suitability of carbon in such repetitious usage was completed, a continuous reduction system was set up, as illustrated in figure 2, which shows the equivalent of a flow sheet of the proposed reduction unit. At the time of this review the construction of this unit (which is also shown photographically in figure 3) had been completed and a trial run begun. Preliminary indications are that the reduction rate will be higher than had been expected from the theoretical equilibrium measurements first mentioned and that certain expected difficulties, such as volatility of the chlorides, were not serious. The maintenance of high gas purity is indicated by this test to be the only major problem involved and may require the development of special methods of testing and analyzing the gas for control purposes. Preliminary estimates further hint at the possibility of producing chemically pure chromium at somewhat less than 20 cents per pound.

5. Thermodynamic Properties of the Hydrates of Calcium Sulphate

The low-temperature specific-heat measurements mentioned under the first item of this report enabled a critical review of thermodynamic data on the various hydrate forms to be undertaken, and intercomparison between physical-chemical measurements in the literature referring to the different forms to be made. While many such physical-chemical measurements are found, this correlation shows existing thermodynamic data to be definitely inconsistent with each other. In spite of the growing industrial importance of gypsum plasters, no significant new work in the thermodynamic properties of these materials has appeared for some 30 years.

It appears that there are at least five forms of calcium sulphate in various degrees of hydration, including the anhydrous, and it is possible that there may be even more. Our chief intent in this problem is to set up a series of thermodynamically authenticated constitution diagrams showing the fields of stability of the various forms and the conditions under which each reverts to another. Such a diagram would have considerable technical and practical value. Thus, from the production end not only would the data help to obtain greater uniformity of product but may also point the way to methods of producing certain forms that are known to have superior quality but can only be manufactured now by expensive methods. Existing dissociation-pressure measurements are definitely inadequate for this purpose, although through the preliminary review that has been made it is possible to approximate the limits of certain of the stability fields.

Further, it is found that the most carefully produced plaster does not always behave satisfactorily "on the job", and the reasons for its sometimes anomalous behavior are little more than occasionally somewhat inspired guesses.

The thermodynamic correlation that has been accomplished in a preliminary way has served to point out the crucial missing data that will enable a satisfactory "constitution" diagram to be presented. They are, specifically, entropy figures for a few more forms of the hydrates and certain heats of hydration or setting. Measurements of both these thermodynamic quantities, designed to supply the crucial missing data, are under way.

Publications

Manuscripts prepared and published during 1936-37

<u>Title</u>	<u>Author</u>	<u>Where published</u>
The Heat Capacities of Molybdenite and Pyrite at Low Temperatures.	C. Travis Anderson	Jour. Am. Chem. Soc., vol. 59, March 1937, pp. 484-487.
The Heat Capacities of Chromium, Chromic Oxide, Chromous Chloride and Chromic Chloride at Low Temperatures.	C. Travis Anderson	Jour. Am. Chem. Soc., vol. 59, March 1937, pp. 488-491.
Progress Reports - Metallurgical Division. 19. Thermodynamic Studies: Calculation of the Specific Heats and Entropies of Metal Vapors from Spectroscopic Data, with Special Reference to Gaseous Iron and Copper.	K. K. Kelley	Bureau of Mines Report of Investigations 3341, May 1937, 19 pp.
Heat Capacities of Selenium Crystals, Selenium Glass, and Tellurium at Low Temperatures.	C. Travis Anderson	Jour. Am. Chem. Soc., vol. 59, June 1937, pp. 1036-1037.

Manuscripts previously prepared but published in 1936-37

<u>Title</u>	<u>Author</u>	<u>Where published</u>
Contributions to the Data on Theoretical Metallurgy - VI. A Revision of the Entropies of Inorganic Substances - 1935.	K. K. Kelley	Bureau of Mines Bull. 394, 1936, 55 pp.
Contributions to the Data on Theoretical Metallurgy - V. Heats of Fusion of Inorganic Substances.	K. K. Kelley	Bureau of Mines Bull. 393, 1936, 166 pp.
Sponge-Iron Experiments at Mococo.	Chas. G. Maier	Bureau of Mines Bull. 396, 1936, 81 pp.
Reduction of Zinc Ores by Natural Gas.	H. A. Doerner	Trans. Am. Inst. Min. and Met. Eng., vol. 121, 1936, pp. 636- 677.

Manuscripts submitted and accepted in 1936-37 but
not yet published

<u>Title</u>	<u>Author</u>	<u>Where published</u>
Chemistry of Anhydrous Chromium Chloride.	H. A. Doerner	Bureau of Mines Tech. Paper 577 (in press).
Contributions to the Data on Theoretical Metallurgy - VII. The Thermodynamic Properties of Sulphur and its Inorganic Compounds.	K. K. Kelley	Bureau of Mines Bull. 406 (in press).
Contributions to the Data on Theoretical Metallurgy - VIII. The Thermodynamic Properties of Metal Carbides and Nitrides.	K. K. Kelley	Bureau of Mines Bull. 407 (in press).

SPECIAL STUDIES SECTION

The special studies section of the Metallurgical Division seeks to advance metallurgical art by discovering and developing radically new processes in milling, smelting, and refining.

Sound Waves for Smoke Settling

The outstanding achievement of the special studies section for the fiscal year is the discovery of the possibility of settling smoke and other fumes and dusts by means of standing sound waves (two waves going in opposite directions, so timed that the points of no movement coincide). Several public demonstrations of this method of flocculating and settling ammonium chloride fumes were staged and have created tremendous interest. The sound waves used for these demonstrations were in the upper region of the audible range and had a frequency of 5,000 to 7,000 cycles per second; they were generated either by Pierce's magnetostrictive oscillator or by a Hartmann air-jet sound generator. Flocculation of ammonium chloride fumes may be accomplished also by using notes of 4,000 to 6,000 cycles per second frequency as obtained from a radio loud speaker, and these instruments may be found suitable for small-scale experiments where great intensity at high frequencies is not demanded.

For practical application of sonic flocculation to the settling of smelter fumes and smoke, the sound generators available are too limited in their power output. Attempts to produce standing sound waves of high intensity are proceeding along with studies on the influence of such factors as frequency and intensity of the sonic oscillations, size, and nature of the suspended particles, as well as composition, temperature, and velocity of the gas by which the fume is carried.

Coercive Force of Ferromagnetic Substances

As a supplement to our previous work on the coercive force of ferromagnetic substances, measurements were undertaken and completed this year on four sets of metal rods that had been subjected to varying degrees of torsion. The objective was to confirm, with greater precision, the straight-line relation between coercive force and specific surface (or grain size) found to hold for magnetite powders. The results were unexpected, in that the curves obtained by plotting torsional work or number of turns against coercive force are linear only through more or less narrow ranges, and, furthermore, the composition of the rods has a determining influence on the course of the curves. The actual experimental work was done by us in the National Bureau of Standards, to whom we are grateful for the continued use of their Ansler-Laffoon torsion machine and their Fahy permeameter, as modified by Sandford, for the measurement of the low coercive forces involved.

Crushing and Grinding

The results of many years of study by John Gross on crushing and grinding have been summarized in a general report, to be Bulletin 402, written on the completion this year of the experiments on explosion shattering.

Publications

A Coercimeter for Magnetically Weak Materials, by E. V. Potter and E. F. Coleman, Review of Scientific Instruments, vol. 7, December 1936, pp. 499-501.

Use of Sound and Supersonic Waves in Metallurgy, by V. H. Gottschalk and H. W. St. Clair, Mining and Metallurgy, vol. 18, May 1937, pp. 244-247.

ELECTROMETALLURGICAL SECTION

Electrometallurgical Laboratory at Boulder City, Nev.

Experience with the 50-kilovolt-ampere electric furnaces at the Reno laboratories of the electrometallurgical section proved that larger units were necessary for studying the electrothermal reduction of low-grade aluminum-potash ores, manganese ores, and other ferro-alloy raw materials.

An appropriation was allotted for the purpose of developing methods for the electrometallurgical application of Boulder Dam power and the utilization of raw materials from that area. A building that had been the garage for The Six Companies was purchased for conversion into an electrometallurgical laboratory in Boulder City, Nev. Figure 4 is an external view of this building soon after its acquisition by the Bureau, and figure 5 is an interior view taken at about the same time.

Work was begun immediately on the conversion of this building into a testing plant capable of handling semicommercial-size electric furnaces. Most of the work at the Boulder laboratory this year has been confined to the erection and installation of equipment. The staff consisted of a metallurgical engineer in charge, a consulting electrical engineer, a chemist, a ceramist, two mechanics and mill-wrights, two electricians, one principal clerk, and three laborers.

The first unit to be installed was a machine shop, where much of the equipment has been made.

An ore-dressing unit capable of handling about 24 tons per day has been installed. Figure 6 shows the rod mill and tables of this unit. Classifiers and screens are on the floor above.

Air-conditioned offices were built in the front of the building for the engineer-in-charge and the principal clerk.

A 3-phase furnace transformer capable of delivering 125 to 1,500 kv-a at 55 to 170 volts was purchased and installed. This wide voltage regulation is desirable because of the varied type of ores to be handled.

A 3-phase 1,500-kv-a arc shaft-furnace was expected to be in operation before July 1937. Figure 7 shows some details of its construction. Most of the parts were made in our shop.

A single-phase coke-resistance furnace for small charges was built and has been in operation on several problems. This draws about 125 kw.

Complete equipment for studying the nature of ceramic materials from this vicinity was erected and has been in operation for most of the year. This includes an 80-kw. Globar resistance furnace, as shown in Figure 8, driers, blunger, pebble mills, potter's wheel, screens, and other machines.



Figure 6.—Ore-dressing unit.

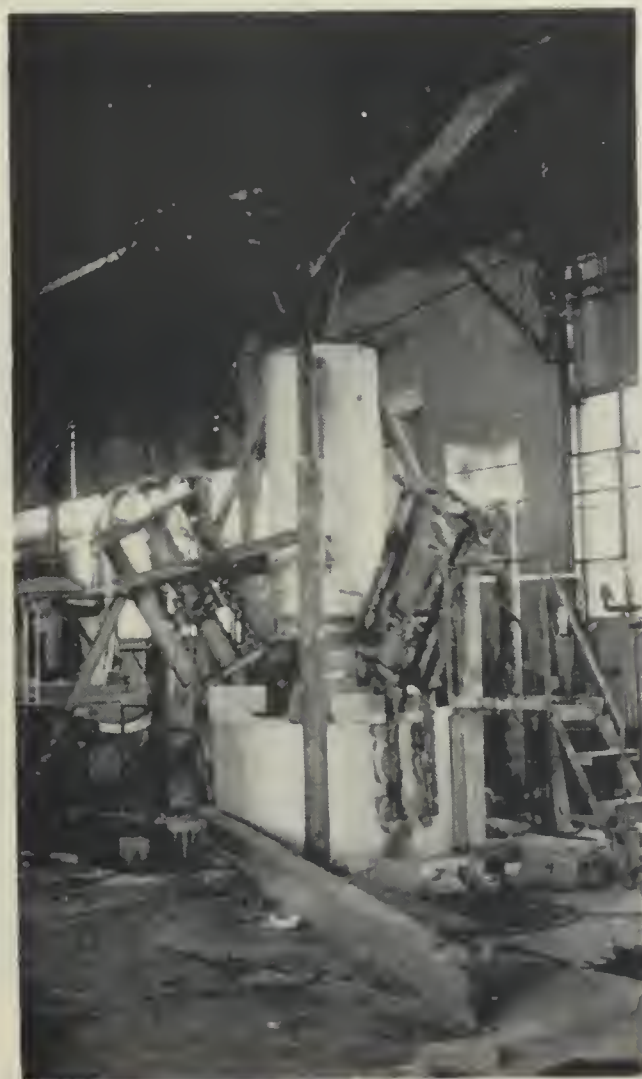


Figure 7.—Three-phase 1,500-kilovolt-ampere arc shaft furnace.



Figure 8.—Eighty-kilowatt resistance furnace for ceramic materials.



Figure 9.—Electric-furnace room at mid 1937.



Figure 10.—The Experiment Station at mid 1937.



Figure 11.—Determining the effect of impurities on the disposition of manganese and methods for purification.

An analytical laboratory with sample equipment to do almost any kind of metallurgical work has been installed.

A general view of the furnace room at the present writing is shown in figure 9.

Landscaping of the grounds and renovation of the building have added greatly to its appearance, as shown in its present state in figure 10.

The total power supplied to the laboratory by the Bureau of Reclamation will be 2,000 kv-a, of which only 250 kv-a has been allowed so far. This amount has been available only since May 10; consequently, it was impossible to carry on much experimental work other than the tests on the ceramic materials.

Electrothermal Studies on Alunite

It was reported last year^{3/} that potassium sulphate could be obtained as a separate phase by the fusion of alunite with borax. During the course of further investigations along these lines, fusions of alunite with boric acid were made in the 80-kw Globar furnace. Sulphur trioxide is practically completely removed during the fusions and leaves a mixture of what appears to be potassium borate, boric acid, alumina, and silica in the form of a milky glass.

This material is leached with water, giving rise to a solution of potassium borate and a residue of Al_2O_3 and SiO_2 . These are filtered off. Sulphuric acid is added to the potassium borate solution and the solution is evaporated. Evaporation produces a mixture of K_2SO_4 crystals and boric acid. It is intended to attempt the separation of these two substances by means of flotation in their own brine. It is believed that the alumina-silica residue can be treated by any of several methods, such as the Bayer or the Hall process, or the alumina may be dissolved in sulphuric acid to produce an aluminum sulphate. Experimental figures to date indicate that evaporation costs may be rather high.

There are considerable tonnages of colemanite in the region around Boulder Dam, enough to supply the demand for borate products for many years. This colemanite is a potential source of boric acid and as such should serve as one of the necessary raw materials in the treatment of alunite. Several fusions of colemanite and alunite gave unpromising results, as none of the fusions were water-soluble. Further work on this problem has been delayed through lack of power.

Preparation of Ferro-Alloys from Domestic Ores

Work on this problem has been confined chiefly to the sampling and analysis of various ores from the vicinity of Boulder Dam because electric furnaces were not available to attempt their smelting.

^{3/} Progress Reports, Metallurgical Division, Electrometallurgical Investigations: Report of Investigations 3322, U. S. Bureau of Mines, 1936, p. 52.

The copper-nickel ores from the Bunkerville district in southern Nevada have been sampled and in general show a combined copper-nickel content of 0.8 to 1.0 percent. A 100-ton sample of this ore is being obtained and will be used in smelting tests in the large 3-phase electric furnace.

A rough table concentrate was made from a sample from this district and was calcined in the Globar furnace. The analysis of the calcine shows a copper content of 3.3 percent and a nickel content of 5.2 percent.

Through a cooperative agreement arranged by the United States Bureau of Mines and the Boulder Dam Manganese Corporation, work has been started on the electrothermal smelting of the manganese ore from the Three Kids district. It has been possible to obtain in the single-phase resistance furnace a manganese metal of the following composition:

<u>Constituent</u>	<u>Percent</u>
Mn	88.10
Fe	3.40
Si	.48
Cu	1.69
C, by difference	6.33

This work was begun just recently and complete data are not yet available.

Investigations of Minerals^{4/} from the Boulder Area

Much of this work has been confined to the determination of the suitability of minerals in the Boulder region for use in the ceramics industry. Thirty samples taken from various localities within the Boulder area were subjected to fusion tests in the Globar furnace. Several excellent kaolins were found, as well as good clays. Several ceramic bodies were made, essentially with material from the Boulder area; an example of one of these bodies is shown in the following table:

<u>Ingredient</u>	<u>Percent</u>	<u>Constituent</u>	<u>Percent</u> (<u>approximate</u>)
California fireclay	50	Al ₂ O ₃	28.4
Goodsprings talc	20	SiO ₂	45.6
Caliente (Nev.) kaolin	20	CaO	.2
Kingman (Ariz.) spar	10	MgO	6.8
	<u>100</u>	Fe ₂ O ₃	<u>1.4</u>
			82.4

^{4/} Reference should be made here to, Hewitt, D. F., and others, Mineral Resources of the Region Around Boulder Dam: U. S. Geol. Survey Bull. 871, 1936, 197 pp.

This body was fired to a white color and would be suitable for making wall tile, art pottery, and stoneware. A second body had the same clays but in different amounts:

<u>Ingredient</u>	<u>Percent</u>	<u>Constituent</u>	<u>Percent</u> (approximate)
California fireclay	30	SiO ₂	49.0
Goodsprings talc	30	Al ₂ O ₃	22.8
Caliente kaolin	20	CaO	.2
Kingman spar	20	MgO	10.1
	<u>100</u>	Fe ₂ O ₃	<u>1.5</u>
			83.6

This body worked well at 1,100° C. but deforms at higher temperatures and was somewhat off color; it could be used, however, for vitreous floor tile and low-fired art pottery.

Electrolytic Manganese

The continuous electrodeposition of high-purity manganese from a sulphate solution is possible, provided that the manganese lost from solution by deposition is replaced by the addition of a manganese compound soluble in the spent electrolyte.

It has been shown^{5/} that pure manganous carbonate added to the spent acidic electrolyte will regenerate manganous sulphate. Preparation of pure manganous carbonate from manganese ore is a somewhat laborious procedure and requires the use of reagents that are not easily recoverable.

The use of dilute sulphuric acid as the leaching agent is desirable because the spent electrolyte after electrolysis of a manganous sulphate solution is dilute sulphuric acid, and its re-use would lead to a large saving in the cost of reagents.

The results of the preliminary work indicated the need for a more direct procedure in separating by electrolytic methods metallic manganese from elements associated with it in domestic ores.

Except for occasional deposits of the minerals rhodochrosite (manganous carbonate) and rhodonite (manganous silicate), manganese is found usually as an oxide or hydrated oxide in the quadrivalent stage of oxidation, which is insoluble in dilute sulphuric acid. Like manganous carbonate, manganous oxide reacts readily with sulphuric acid to form manganous sulphate.

Clevenger and Caron^{6/} showed that manganese dioxide contained in ores could be reduced to manganous oxide by calcining in a reducing atmosphere at 600° C.

^{5/} Reference 3.

^{6/} Clevenger, Galen H., and Caron, H., The Treatment of Manganese-Silver Ores: Bull. 226, Bureau of Mines, 1925, 110 pp.

It is evident that manganous oxide contained in reduced ore can be used instead of manganous carbonate as a raw material for electrolysis, provided the leach solution can be purified before electrolysis.

With funds provided by a cooperative agreement with the Western Minerals Exploration Co., an experimental plant was erected for the production of electrolytic manganese from domestic ores. Ore from the Three Kids mine in Nevada, 7 miles from Boulder Dam, was selected for test purposes. It is essentially a wad or bog type manganese ore of the following analysis:

<u>Constituent</u>	<u>Percent</u>	<u>Constituent</u>	<u>Percent</u>
Mn	32.10	As	0.13
Fe	.68	SiO ₂	17.38
Pb	1.54	CaCO ₃	.09
Cu	.35	MgCO ₃	3.56

A rotating kiln-type furnace was erected. The furnace was essentially a 6-inch rotating pipe 16 feet long and sloping $3\frac{1}{8}$ inch per foot. The upper 8 feet was heated externally with an oil burner, and the lower 8 feet was cooled with a water spray. Ore mixed with bituminous coal was fed into the hot end. The high temperature liberated the volatile constituents of the coal, which, in turn, reduced the manganese in the ore to manganous oxide. The ore was cooled in a reducing atmosphere and removed from the cold end of the furnace. The capacity of the furnace was about 1 ton in 24 hours.

The ore was leached in a 24-gallon oak barrel provided with a stirrer of wood. Enough sulphuric acid was added to the ammonium sulphate solution of 190 grams per liter concentration to obtain a manganous sulphate content of 50 grams per liter, and ore was added to the acid solution until the solution became neutral. Ninety minutes was required for each leaching operation.

The leach solution was brought into contact with iron nails for copper removal and to increase the iron content for arsenic removal by subsequent oxidation and precipitation of ferric iron from the solution. The purified solution was filtered and pumped to the cathode feed tank for electrolysis.

The electrolytic cell consisted of eight mild steel cathodes and nine lead anodes separated by storage-battery wood diaphragms. Each cathode had a plating area of 1.63 square feet, and a current density of 40 amperes per square foot at 5 volts was used.

The spent electrolyte, which contained 30 grams per liter of sulphuric acid, was used as a leaching agent for additional reduced ore. The solution was again purified and electrolyzed.

The total of 7 pounds of high-purity manganese was produced. The current efficiency was between 35 and 40 percent. Because of a definite resolution tendency, it was necessary to remove and strip the cathodes after 12- to 24-hour plating periods.

The results obtained established the possibility of producing electrolytic manganese from domestic ore by a direct and regenerative process that would require reagents only to replace leaching and mechanical losses in the circuit.

The results failed to establish a satisfactory current efficiency and a more desirable 48-hour plating period. The trouble has been traced to impurities in the solution. Methods have been studied and developed for additional solution purification.

On the basis of experience gained in operating the small plant, a new plant has been designed and constructed. The objective is to produce electrolytic manganese on a simulated plant scale by using four cathodes of 7.5 square feet plating area each; to increase the current efficiency and plating time by greater purification of the electrolyte; and to reduce the electrode potential from 5 to approximately 4 volts by using canvas bags as diaphragms in place of the wooden storage-battery separator.

The control of impurities in the electrolyte during electrodeposition of manganese is of primary importance. Studies of the effect of different impurities and methods for purification are being made in the laboratory cell shown in figure 11.

The electrolyte is fed into the cathode compartment from container A. Metallic deposition takes place and the spent catholyte flows into the anolyte compartments, then leaves cell B through the launder, to filter C. Some suspended manganese dioxide is formed by anodic action and is removed from solution by the filter. A portion of the filtered spent electrolyte is recirculated through the anode compartments to sweep out the manganese dioxide and prevent settling and clogging within the cell. The remainder of the spent anolyte flows into container D to be used for leaching additional ore or to be discarded. Other containers shown in the photograph are for storage.

The cell, filter, and containers are lead-lined with 99.99+ percent pure sheet lead. Anodes are lead of equal purity. Seventeen-percent chromium steel is being used for cathode material. Cathode plating area is 0.5 square foot.

The cell was operated 302.5 hours during May in 24-, 16-, and 12-hour and shorter plating periods.

Tentative conclusions regarding impurities are:

1. Arsenic concentration can be reduced to less than 0.00005 g/l, copper to less than 0.0005 g/l, iron to less than 0.001 g/l, and molybdenum to less than 0.00001 g/l conveniently and inexpensively. No conclusions have been reached regarding zinc removal.

2. Phosphorus, as phosphate, is an objectionable impurity. It is removed during leaching as an iron phosphate or as magnesium ammonium phosphate.

3. Impurities such as bismuth, tin, silver, and antimony found in small quantities in fertilizer-grade ammonium sulphate produced as a by-product of western smelter gases, but so far undetected in domestic manganese ores, interfere with electrolysis. It is believed that a protracted study of the removal of these impurities is not justified.

Magnesium from Magnesite Ores of the Northwest

After an intensive study of all the methods for producing metallic magnesium that have been found in the literature, the following conclusions have been reached: On the basis of information now available, there are just two methods by which it may be profitable to extract magnesium from magnesite ores—chlorination of technically pure MgO by heating with carbon and chlorine, followed by electrolysis of the fused chloride, and direct electrothermic reduction of calcined magnesite by carbon, the product being purified by redistillation.

Reduction of magnesia by other metals, alloys, or hydrogen is not feasible either for economic or for thermodynamic reasons. Chlorination and electrolysis of the chloride comprise an indirect "oxide" process that is more efficient than any electrolytic method that has been reported. Direct electrolysis of the oxide, dissolved in a fused salt, has not been successful but may have possibilities. Electrothermic reduction by carbon is the simplest and most logical method for magnesite. It appears to have unique advantages.

For the purpose of studying the electrothermic reduction of magnesia by carbon, an arc furnace was designed and constructed to permit working under reduced pressures.

The reaction chamber is cylindrical, 10 inches in diameter and 10 inches high. The walls are constructed of granular, dead-burned magnesite bonded with corn syrup and caustic-calcined magnesite. The roof is made of 1-1/2-inch graphite "beams" laid closely and covered with loose magnesite. The refractory is inclosed in a gas-tight iron housing.

The electrodes are mounted horizontally so that the arc is centered 3 inches above the bottom of the reaction chamber. They are mounted on adjustable electrode holders and project through insulated stuffing boxes on the furnace shell. Both the holders and stuffing boxes are water-cooled.

In the same horizontal plane as the electrodes, on an axis at 90° from them, are two ports or openings. Through one of these, 1-1/2 inches in diameter, a screw feed pushes cylindrical briquets of MgO -carbon mixture into the reaction zone. The other port, 1 inch in diameter, leads to a water-cooled pipe connected to a cyclone dust collector and bag filter. Inlets for the cooling hydrogen gas are provided in this water-cooled port.

In the bottom of the reaction chamber are laid broken pieces of graphite intended to serve as a resistor bed upon which the reaction takes place. After the furnace is hot and the electrodes are pulled apart, more or less of the current arcs through this resistor bed.

Hydrogen is admitted through a flowmeter, and the gas exit from the bag filter also passes through a "low-resistance" flowmeter. These two meters are adjusted so that the readings for the same flow of gas correspond. Thus, the difference between the two meter readings is a direct measure of the rate at which the reaction is taking place.

A short run was made, in which 1 kilogram of ore-carbon briquets was fed during a half-hour period. This was done before the refractory was completely degasified to prevent overheating and to obtain information on the behavior of the reaction. The operation appeared to be smooth and satisfactory. Each intermittent feed of the mixture caused a marked increase in the evolution of gas, and it appeared that the rate of feed did not nearly equal the capacity of the furnace.

After the furnace had cooled, it and the condensing system were opened and inspected. The retort walls and roof were in excellent shape. About one-third of the material fed had not reached the hottest part of the reaction zone and did not react. A considerable portion of the charge had been deposited (by reversal of the reaction) upon the walls of the retort.

There was not enough distillate in the condensing system for satisfactory estimation as to quantity. It consisted of MgO and carbon. Failure to obtain metal was not unexpected, because of the short duration of the run, the small amount of hydrogen used, and the large amount of moisture evolved and condensed. Any magnesium that may have condensed as metal would be pyrophoric and quickly oxidized in the presence of moisture or air. Although the condenser was dry at the end of the test, about half a pint of water collected as the furnace cooled. The moisture must have come from the furnace lining, and good results are not possible until it is eliminated.

The feeding device was designed to feed only cylindrical briquets at a slow rate. This is being replaced by a plunger-type feeder that will deliver loose powder at a faster rate. No other mechanical change is contemplated at present.

An apparatus was designed to study the production of anhydrous magnesia chloride by treating magnesia and carbon with chlorine at elevated temperatures. Chlorine was obtained from an electrolytic home-made cell. Pure MgO , CaO , and various grades of magnesite ores, both raw and calcined, were tested. In most cases a high fraction of the material was converted to anhydrous chloride. Best results were obtained at a temperature just below the melting point of $MgCl_2$, $712^{\circ} C$.

The tests have shown that chlorination is not difficult, but this step cannot be used to make a separation of Mg from Ca . Therefore, to make a satisfactory electrolyte the raw material must be freed from Ca .

A series of roasting and leaching tests was made on crude magnesite in an attempt to produce selective calcination that would result in a mixture of MgO and $CaCO_3$.

Results of considerable interest have been obtained. Between 650° and 700° C., magnesite content of a crude ore can be calcined completely without appreciable changes in the calcite, dolomite, or serpentine. The calcined magnesite crumbles easily and can be separated from the other constituents by classification, wet or dry. This method probably will produce a better separation than is possible by flotation.

Comprehensive data have been obtained on separation made by elutriation of calcined crude magnesite. In a representative composite test, the calcined concentrate analyzed 87.23 percent MgO, 0.37 percent Ca, 8.25 percent SiO₂, and 4.15 percent R₂O₃. Recovery of total MgO was 67.4 percent. Most of the MgO lost was present as unrecoverable serpentine.

Flotation tests also have been made on some of these calcines, generally with remarkably good results. In one case the insoluble content of the raw ore was reduced from 7.2 percent to 0.9 percent, and a recovery of 84.7 percent of the magnesia was obtained.

PUBLICATIONS

<u>Title</u>	<u>Author</u>	<u>Where published</u>
Studies on the Treatment of Domestic Chrome Ores	J. Koster	Report of Investigations 3322, Progress Reports, Electrometallurgical Section, Metallurgical Division, Bureau of Mines, 1936, 64 pp.
Electrolysis of Manganese Solutions	S.M. Shelton	
Electrometallurgical Studies on the Treatment of Alunite	R.G. Knickerbocker and J. Koster	
Electrolytic Manganese	J. Koster and S.M. Shelton	Eng. and Min. Jour., vol. 137, 1936, pp. 510-513.
Present Outlook for a Magnesium Industry in the Northwest and a Discussion of Methods by which Magnesium Metal may be obtained from Magnesite Ores.	H. A. Doernor	Bulletin P, Mining Experiment Station, State College of Washington, July 1937, 90 pp., including bibliography of 447 references.

ORE-DRESSING SECTION

The work of the ore-dressing section comprises studies of ore-dressing processes. It is concerned mainly with such basic principles as may determine the efficacy of old methods and upon which new processes may be developed.

The problems selected are preferably of general rather than purely local interest; consequently, the ores used in the investigations represent types, not specific deposits.

The following outline renews the problems that have received special attention during the past year:

Grinding Studies

Roll-grinding with two sets of 54- by 24-inch rolls in parallel was examined. The new feed was as coarse as 1 inch, and the finishing was dry through 10-mesh screens. The rate of finishing was as high as 100 tons an hour. The objective was to learn if the advantage of heavy circulating loads, as claimed for ball mills, applies to rolls.

In the first analysis it is seen that the deportment of rolls is different. When the feed to rolls increases, the power increases, but with ball mills the power remains about the same. The ball mill applies less energy to a unit of ore when the feed is heavy. This gives an opportunity to take finished material out before it is overground. Such an advantage is not discernible in rolls while the spacing is unchanged.

The conclusion was reached that a heavy circulating load on rolls does not increase the selective grinding and efficiency.

The efficiency of rolls was compared with ball mills in the same plant. It was found that fundamentally rolls and ball mills have about the same efficiency in their respective positions in the flow sheet and that increasing the load to rolls where the setting remains unchanged did not increase the selective crushing.

In closed-circuit ball-milling it has been shown by tests that heavy circulating loads give the least spread in particle sizes between feed and discharge and light circulating loads give the greatest spread in particle size.

Obviously, the appropriate ball size may be selected more exactly when the spread of particle size from end to end of the mill is a minimum, and if we fail to endeavor to use the correct ball size while seeking a heavy circulating load we neglect to apply an advantageous principle.

Two investigators in ball-mill grinding by laboratory methods have reported what seemed to be anomalous results when the circulating load reached

a high point: They stated that the output dropped off when the circulating load reached a certain high amount, depending on the particular ore. This subject has been investigated in closed-circuit batch-grinding by running the circulating load up as high as 33 times the original feed. The statements of those investigators could not be confirmed because the output increased regularly as the circulating load was increased.

A conical mill 6 feet long, 2 feet in diameter at the feed end, and 1 foot in diameter at the discharge end worked with good efficiency but was unsatisfactory in capacity. The capacity did not justify the floor space occupied. The small end seemed to be almost useless. The failure of the small end was due to the pyramiding effects of two causes - its small diameter and its low critical speed when the big end was run at maximum speed.

Calculations showed that the last unit section had a capacity of only 12 percent of that of the feed end.

The effect on grinding capacity, when finished material remains in the circulating load, has been the subject of conjecture, but it is believed that heretofore the effect has not been determined experimentally. By experiments it was found that the percentage of decrease in mill capacity is almost identical with the percentage of finished material in the composite feed. For example, when 40 percent of the finished material was returned and it amounted to 19.1 percent of the composite feed, the reduction in capacity was 21.5 percent. The ultimate grinding, however, was not changed materially because the finished material is reduced further when it is returned to the mill.

The unfinished mill effluent had about the same sizing analysis throughout a wide range in amount of finished material being returned. In terms of plant operation, these results mean that when the percentage of circulating load is set and the classifier sends an additional amount of finished fines back to the mill, the amount of new feed has to be reduced in proportion to the additional fines in the composite feed.

In the selection of balls in grinding, the perfect spheres have been found best. An accumulation of balls with distorted shapes due to tumbling or erosion may reduce the capacity as much as 7 percent.

The Bouyoucos hydrometer has been examined and accepted as a device for finding the sizing analyses of subsieve sizes. The method, which is based on sedimentation principles, was developed by Bouyoucos for determining colloids in soils. When the sample is prepared, hydrometer readings are taken at intervals. Corrections are made for specific gravity, temperature, and other factors. From these data a complete sizing analysis down to 1 micron can be made.

The coercimeter has proved to be a valuable accessory in estimating the results of grinding. It reacts only to magnetite or a similar mineral; therefore, the ore ground in the tests has to be magnetite. Whereas much time was consumed by the old methods in determining surface of ground products,

now magnetite is chosen as the ore in the testing and the relative amount of surface in the products from several tests can be determined in a few minutes.

PUBLICATIONS

<u>Title</u>	<u>Author</u>	<u>Where published</u>
Ball-Mill Grinding	W. H. Coghill and F. D. DeVaney	Bureau of Mines Bull. 402 (in press).
Relation of Ball Wear to Power in Ball Milling	W. H. Coghill and F. D. DeVaney	Eng. and Min. Jour., vol. 138, July 1937, pp. 337-340
Use of the Coercimeter in Grinding Tests	F. D. DeVaney and W. H. Coghill	To be paper of A.I.M.E.
Observations from Some Crushing and Grinding Tests	W. H. Coghill and F. D. DeVaney	The Mines Magazine, vol. 27, June 1937, pp. 7-10.
Sub-Sieve Size Distribution Follows Same Law as for Coarser Particles	Fred D. DeVaney and Will H. Coghill	Rock Products, vol. 40, Oct. 1937, pp. 58-59.
A Short-Column Hydraulic Elutri- ater for Subsize Sizes	S. R. B. Cooke	Report of Investiga- tions 3333, Bureau of Mines, February 1937, pp. 39-51.

Iron Ores

For shower roasting of hematite a gran-annular furnace was rebuilt so that the ore could be dropped through a central vertical shaft lined with graphite.

An iron ore through 65-mesh is reduced effectively at temperatures of 750° to 900° C. during a passage in the furnace of about 0.2 second. Material coarser than 65-mesh is not sufficiently reduced to permit low intensity or a.c. magnetic concentration. Whereas, the tests have shown that temperatures exceeding 950° C. cannot be used without danger of forming concretions, it seems evident that to procure effective roasting of material coarser than 65-mesh the period in the furnace must be increased.

A high-intensity magnetic separator was constructed, based on the principle developed by G. Granigg of Leofen, Germany. This separator was designed to handle either wet or dry material and to have a field of about 12,000 oersteds. Tests on a dry limonite and barite ore gave clean limonite concentrates, but to procure low residues it was necessary to retreat the tailings several times. The effectiveness of the separator with a wet pulp has not yet been checked.

Chemistry of Flotation

During the past year an attempt has been made to obtain quantitatively some information on the deportment of silica in abstracting certain wetting agents from solution. The abstraction of the reagents was estimated in terms of their nitrogen (NH_3) content by using the micro-Kjeldahl method.

The work was started with apparatus of crude design. This was satisfactory for preliminary work but later apparatus of improved design was obtained. Several methods of procedure in the abstraction tests were tried; the one finally adopted is, briefly, as follows:

One hundred grams of clean quartz is agitated with 200 ml of distilled water and a known amount of reagent. The pulp is filtered and samples of the filtrate are taken for analysis. The difference between the quantity of reagent added and that remaining in the filtrate is the amount abstracted by the quartz.

Tests were made on the following reagents: Emulsol 660-B, duPont Q and 243, DLT 521 and 57-A, tetra-ethyl-ammonium chloride, and two dyes, crystal violet and toluidine blue. The tests with Emulsol 660-B and tetra-ethyl-ammonium chloride failed because they could not decomposed satisfactorily to liberate the nitrogen as ammonia. The other compounds worked satisfactorily, the data yielding abstraction curves of the adsorption type.

Other tests indicated that the abstraction of potassium octyl xanthate from flotation solution for varying periods of treatment was a reaction of the adsorption type. The xanthate used had been prepared from commercial octyl alcohol, which was apparently a mixture of long-chain alcohols. A single isomer was desirable; hence, the alcohol was fractionated and yielded about 30 percent of normal octyl alcohol. This alcohol was used to prepare the potassium xanthate for continuing the investigation.

The preliminary tests had been made on Tri-State sphalerite that had been activated by lead or copper salts in the ore. To avoid this activated condition, a jig concentrate was obtained from Mascot, Tenn. This was cleaned by repeated classification and tabling after crushing in rolls between each cleaning operation. Three products were obtained, -28 + 65, -65 + 200, and minus 200-mesh. As a final cleaning operation, the different products were passed over a high-intensity magnetic separator. Unlike the Tri-State ore, this sphalerite had not been activated for flotation, as was indicated by its nonfloatability with ethyl xanthate. However, it was floated readily with octyl xanthate without activation.

In the first new series of abstraction tests the period of treatment was 1 minute, and the quantity of xanthate was varied. As the quantity of xanthate increased from 0.01 to 0.03 milli-equivalent per 200 grams of mineral (65 to 200-mesh), almost all the xanthate was removed from the solution by the mineral; and as the xanthate was increased further to 0.30 milli-equivalent per 200 grams of mineral, xanthate appeared in the various

filtrates. There was a deficiency, however, and it amounted to the maximum abstraction mentioned above. This bears out the preliminary tests in that the removal of xanthate approaches a maximum that is independent of the quantity of reagent.

Difficulty has been experienced in the iodimetric determination of xanthate in an alcoholic solution. This has been avoided partly by leaching, so that only a small quantity of xanthate can report in the alcohol filtrate, thereby minimizing any titration error. Much time has been required to work out satisfactory methods of titration.

The second series of tests, in which the treatment period was increased to 4 minutes, has been started.

Flotation of Nonsulphides

Complex Molybdenum-Vanadium Ores

The laboratory results on this investigation have been applied to a plant that has been in successful operation for the last year at Mammoth, Ariz. The laboratory work has been published under the title "Flotation of Complex Molybdenum-Vanadium Ores from Mammoth, Ariz." and appears on pages 21-38 of Report of Investigations 3333 of the United States Bureau of Mines, 1937.

Talc-Magnesite Ores

A pilot plant is being operated successfully at Johnson, Vt., in which a flotation separation is being made between talc and magnesite by a plan developed at the Rolla station. High-grade talc concentrates and low-talc magnesite tailings have been produced without difficulty, but the capacity of the plant is low. The operation brings out the need of flotation reagents suitable for finely ground material. The results of the laboratory work were published under the title "Flotation of Vermont Talc-Magnesite Ores," Report of Investigation 3314 of the U. S. Bureau of Mines, 1936, 12 pp.

Langbeinite

Langbeinite, a potash-bearing mineral, was floated successfully in the laboratory, and a paper, "Flotation of Langbeinite from the Potash Field of New Mexico and Texas" (Report of Investigations 3300, U. S. Bureau of Mines, 1936, 4 pp.), was published.

New Reagents

A preliminary report covering the early work on the use and application of wetting agents in flotation has already been published as a section in Report of Investigations 3333, February 1937, pages 3-20. Since this publication, the study has been carried forward and more than 100 wetting agents have been investigated for flotative properties. Although time has

not permitted a thorough study of each particular reagent, the outstanding merits and properties have been found by flotation tests on both sulphide and nonsulphide ores. Nearly all of the wetting agents investigated have pronounced frothing properties and may be substituted for the customary frothers in flotation. Many of the reagents promote flotation of the sulphides and when used in conjunction with addition agents give results that compare favorably with those obtained by xanthate or acrofloat.

Wetting agents may be classified in two groups, depending on whether they yield positively or negatively charged ions in solution. Those reagents so composed that they ionize to give negatively charged ions are often selective collectors for the nonsulphides and enable their flotation from a siliceous or calcareous gangue. On the other hand, those reagents that ionize positively are selective with regard to silica and enable flotation of quartz and certain of the silicate minerals from other nonsulphide ores. These reagents have given satisfactory separations of siliceous minerals from limestone, barite, fluor spar, magnetite, and ilmenite ores. In many instances the separation obtained by floating the silica is superior to that obtained by the customary method of floating the other nonsulphide minerals.

Although the flotation of quartz from ores containing a variety of silicate minerals is usually difficult, the use of suitable addition agents often will permit satisfactory separations. The separation of kyanite and quartz is not difficult. Either of the minerals may be floated selectively with equally good results, depending on the condition chosen for the trial. The flotation of mica from siliceous ores and the separation of the feldspar from the quartz also has been accomplished by control of the pH and the use of suitable addition agents.

The collecting properties of the wetting agents for silica vary considerably and depend on both the ionic grouping and length of the hydrocarbon chain. A series of alkyl pyridinium salts and a number of ninol derivatives have been prepared and tested to determine the effect of the halogen radical and the length of the chain on flotative properties. The results indicate that there is a critical chain-length of maximum collection for each type of compound. The various halogen derivatives appear equally effective.

Correlation of the data obtained on various types of reagents has pointed the way toward cheaper silica collectors. Although certain of these reagents are less effective and lack the specific collecting properties for quartz exhibited by lauryl pyridinium iodide, they possess the advantage of cheapness. Control of the pH and the use of addition agents has permitted satisfactory flotation on several ores. These reagents also appear promising as agglomerating agents. The lipophilic coating induced by their use permits selective oiling and agglomeration of silica. Promising results have been obtained on the separation of chert from limestone in a cement rock. The application of agglomeration of silica to other nonsulphide ores is now being investigated.

ORE-TESTING SECTION

During the second year of its existence the ore-testing section has actively prosecuted work in the standardization of testing methods, in conducting routine tests and analyses for other Government agencies, and in the determination of recommended processes for the treatment of representative ores from mining districts.

Standardization of Testing

The first annual report^{7/} of the ore-testing section, which was published in January 1937, contains a rather comprehensive discussion of ore-dressing tests and their significance. It describes the equipment and procedure used in our laboratories for the examination and testing of ore samples to determine their amenability to beneficiation.

The following subjects were included: crushing and sampling, sizing tests, float-and-sink tests, microscopic study, magnetic tests, heat treatment, agglomeration and tabling, explosive shattering, gravity concentration, and flotation tests, including open- and closed-circuit tests, together with a discussion on the significance of test results and their adaptation to suitable flow sheets.

This report also contained a chapter on the analysis of molybdenum.

Additional results now on hand are being used for the preparation of the second annual report of the ore-testing work, which will include chapters on the testing of gold and silver ores, the use of the spectrograph in ore testing, the determination of molybdenum, the analysis of chromite, the electrodeposition of cobalt, and a rapid method for the identification of feldspars.

The section on testing gold and silver ores will include details regarding the procedures used in our laboratories for the investigation of various types of ores and will discuss such processes as amalgamation, cyanidation, roasting, and flotation.

The use of the spectrograph is finding its place as an essential tool in the field of ore testing. With it the presence or absence of small amounts of valuable elements and of interfering substances can be determined readily.

A new method for the determination of molybdenum in ores and metallurgical products has been developed, which it is hoped may supply a need that has existed throughout the mining industry for an effective evaluation of low-grade products.

^{7/} Davis, C. W. and staff of Ore-Testing Section, Progress Reports - Metallurgical Division, 16. Ore-Testing Studies: Report of Investigations 3328, Bureau of Mines, 1937, 161 pp.

The necessity for an investigation of methods for the determination of chromium in ores and concentration products arose when it was found that usual procedures for this analysis failed to give results sufficiently reliable to permit a reasonably accurate computation of the relative amounts of chromite present in products resulting from beneficiation tests. It has been found that a reoxidation step is necessary in the determination of Cr_2O_3 to insure accurate results for the analysis of ores and concentrates. The dichromate method previously used is unsatisfactory, but a modified permanganate method has proved to give excellent results.

To facilitate the study of ore-dressing possibilities of a cobalt ore it was necessary to develop a relatively rapid method for its analysis. The problem was solved by finding conditions under which the cobalt could be determined electrolytically. In brief, the procedure consists of removing interfering elements with hydrogen sulphide, then with ammonium hydroxide and oxalate. The electrolysis takes place in an ammonical solution in the presence of ammonium sulphate and ammonium bifluoride.

The investigation of possible methods for the beneficiation of ores containing quartz and feldspar has been greatly aided by the discovery that feldspars can be differentially stained by certain organic dyes. This makes possible a rapid identification and rough approximation of the relative amounts of feldspar and quartz present in a sample by simple microscopic examination of the stained products.

Service Tests

This function of the ore-testing section is now recognized as one of great importance, especially in its relation to the needs of the other sections of the Metallurgical Division.

Additional equipment and improvements in present machines and instruments have increased the efficiency and capacity of the ore-testing units. The new, small, laboratory flotation machine that has been constructed permits the handling of small samples such as are obtained as rougher concentrates from moderate-sized tests on lean ores. This eliminates the necessity for large-scale tests of such ores and results in the saving of much time and material in the investigation of treatment methods of low-grade ores.

A belt-type, a.-c., magnetic separator constructed along the lines indicated by the results obtained from the operation of the model machine developed in the special studies section has permitted the testing of ores as to their amenability to treatment by this new type of machine.

The addition of a rotating half-cylindrical sector to our spectrographic equipment enhances its value as an analytical tool, in that quantitative measurements may now replace the rough qualitative approximations of certain elements that were obtained heretofore.

The facilities for spectrographic, microscopic, and analytical testing are being used freely, and the assistance rendered has contributed considerably to the effective prosecution of many problems. About 1,000 tests per month are required to keep up with the demand for this type of work.

The tests are of a widely divergent nature and some of them require unusual technique and special equipment.

Establishment of Ore-Testing Procedures

The work relating to this objective has been continued at a rate commensurate with the available facilities. Many of the samples submitted could be treated satisfactorily by regular methods.

Cyanidation

As a rule, the gold and silver ores could be treated readily by amalgamation or cyanidation. Examples of this are afforded by a sample of black-sand concentrate from New Mexico, from which amalgamation suffices to recover all but a trace of the gold, and by a siliceous gold ore from Washington, in which 95 percent of the gold can be recovered by cyanidation.

In some cases it may be advantageous or necessary to resort to flotation either to remove cyanicides prior to cyanidation or to produce enriched concentrates that can be shipped to a smelter. Such a combined method was found to be effective in the treatment of an ore from Nevada. Flotation of minus 65-mesh material permitted the recovery of 70 percent of the precious metals in a high-grade concentrate, and cyanidation of the tailings increased the recovery to about 92 percent.

Flotation concentrates may be rendered susceptible to concentration in some cases by subjection to a roasting treatment. In the presence of lead the roasting temperature must be kept at a minimum, and if cyanicides are produced it may be necessary to wash prior to cyanidation.

Long-continued grinding has been found essential to the satisfactory recovery of precious metals from certain refractory ores by cyanidation; this is particularly important for the treatment of ores containing tellurides and for oxidized ores containing silver; excessive grinding, on the other hand, may not result in optimum recovery.

Magnetic separation

Many tests of ores were made to determine whether their beneficiation might be effected by magnetic separation. In most cases pretreatment by roasting of the samples was required. The flash-roasting procedure developed in the ore-dressing section is especially effective for this type of treatment and doubtless will make it practical, as this roasting is completed rapidly and economically.

Magnetite, manganese, limonite, ilmenite, and nepheline syenite ores were found to respond to some type of magnetic separation.

Magnetite can be treated easily by direct magnetic separation with d.-c. machines.

In certain cases manganese ores can be freed from excess iron minerals to produce high-grade concentrates. An illustration of this is the method developed for treating products now wasted at a plant near Embreeville, Tenn., that will permit the recovery of good grade of manganese concentrate and a fair grade of manganiferous iron ore. This method consists in tabling the classified feed to reduce silica, subjecting the resultant products to a magnetic roast, and finally in passing the calcine over a.-c. magnetic separators. The recovery of the manganese from the roasted product is practically complete. Flotation and gravity treatments failed to give a satisfactory solution to this problem.

Limonite ore also is concentrated readily by a.-c. separation if it be subjected first to a magnetizing roast. In some instances enough organic matter is present so that this roast can be made with a limited amount of air, rather than resorting to roasting in a reducing atmosphere.

Certain ilmenites in which the magnetite was removed by d.-c. separators failed to produce high-grade titanium products. Ilmenite products from gravity concentrations were found to be susceptible to grading up by subjecting them to a magnetizing roast followed by a.-c. separation.

The iron content of a nepheline syenite ore was reduced from 3.29 percent to 0.41 percent Fe_2O_3 by the use of high-intensity induced roll-type separators.

Flotation

Flotation tests of sulphide ores presented no outstanding problems, there being little or no variation from standard practice required to give satisfactory results.

In general, the ore was crushed to minus 10-mesh and then stage-ground to a size at which mineral liberation was accomplished. Distilled water was used for grinding and for flotation. As an example of this type of treatment, a molybdenum ore from Colorado containing 2.8 percent molybdenum was ground to minus 200-mesh; the molybdenite was readily floated with steam-distilled pine oil, using a limited amount to retard the flotation of pyrite. By repeated cleaning and the use of small quantities of sodium silicate, any desired grade of concentrate could be obtained. Extraction of molybdenite was 92 percent.

Complex sulphide ores require a differential treatment to separate and recover their valuable constituents.

A lead-zinc ore from Montana contained galena, sphalerite, pyrrhotite, chalcopyrite, quartz, siderite, barite, and mica. This ore contained 2.6 percent lead and 1.3 percent zinc. Mineral liberation was effected at minus 150 mesh. Lead flotation concentrates assaying 6.7 percent lead and 2.6 percent zinc with a recovery of 92 percent were produced by the use of the following combination of reagents: Sodium carbonate, sodium silicate, potassium cyanide, zinc sulphate, ethyl xanthate, and copper sulphate.

In attempting to float nonsulphide minerals from certain products resulting from gravity concentration, difficulties were encountered due to change in mineral surfaces during pretreatment to the presence of colloidal matter in slimes that coated the mineral grains, consumed the collector reagent, and floated to contaminate to concentrate. The difficulty has been overcome by the use of portions of head samples for making tests.

Some success was attained in the flotation of other ores containing one or more of the following minerals: Barite, magnetite, martite, ferberite, scheelite, and cassiterite.

In some cases, oxidized ores may be sulphidized by a preliminary treatment and then subjected to flotation. Examples of cases in which such a treatment has proved practical include a calamine-cerussite ore, from which a good grade lead concentrate and a zinc concentrate of acceptable grade were obtained, as well as a jarosite ore and a cobalt ore containing stainerite.

NONFERROUS METALLURGY SECTION

Microscopy of Ores

As mentioned in the foreword, a new Panphot microscope was installed early in 1937 in the laboratory at Salt Lake City. With its many interchangeable parts, it is possible to study any mineralogical occurrence by lighting in various ways or by etching without disturbing the bodies or particles being examined. Some examples of the type of assistance that is supplied by the use of this instrument are shown in figures 12-15. This microscopic equipment has permitted the examination and testing of ores and metallurgical products for other sections of the Metallurgical Division and thus act as a service unit. The identification of mineral constituents - the determination of their physical condition, chemical alteration, and their association with other minerals - has been of great assistance in solving the various problems under investigation.

The study of the silver minerals of the Tintic Standard and other ores of the district has been greatly accelerated by using this microscope. It has been found that the forms in which silver was being lost from a large sample of flotation tailing made by the best procedures known at the start of the investigation were about 75 percent as anglesite and cerussite and considerable as metallic silver. The occurrence of silver in oxidized lead minerals as finely divided metal appears to be a new observation. About 20 percent of the silver was lost as argento-jarosite surrounded with limonitic material. Some 5 percent of the silver was lost in minerals of the tetrahedrite family or alteration products thereof. Most of the anglesite and cerussite was liberated by grinding to 150-mesh. It was also learned that both the anglesite and cerussite contained bands of light and dark color. The light bands contained less silver than did those of darker color; the light-colored anglesite contained 19.7 ounces of silver per ton and that of dark color 48.4 ounces; the light cerussite contained 32.7 ounces of silver and that of dark color 81.5 ounces. The particles of metallic silver in the lead minerals were very small, being from 20 to 5 microns in size. This peculiar occurrence of silver in oxidized lead mineral was noted first in rich lead specimens. When samples of oxidized minerals were panned from flotation tailings and briquetted, the same occurrence of silver was found along with some tarnished silver particles coated with films of lead sulphate and other slimes.

Since all attempts to float the gold and silver from the semioxidized siliceous silver-lead ores from the Tintic district proved ineffective, it has been proposed to float as much of the oxidized lead and silver as possible and send it to the lead smelter, while the tailing containing the rest of the silver and a little oxidized lead could be shipped to a copper smelter, as the rates of the latter are less than of the lead smelter.

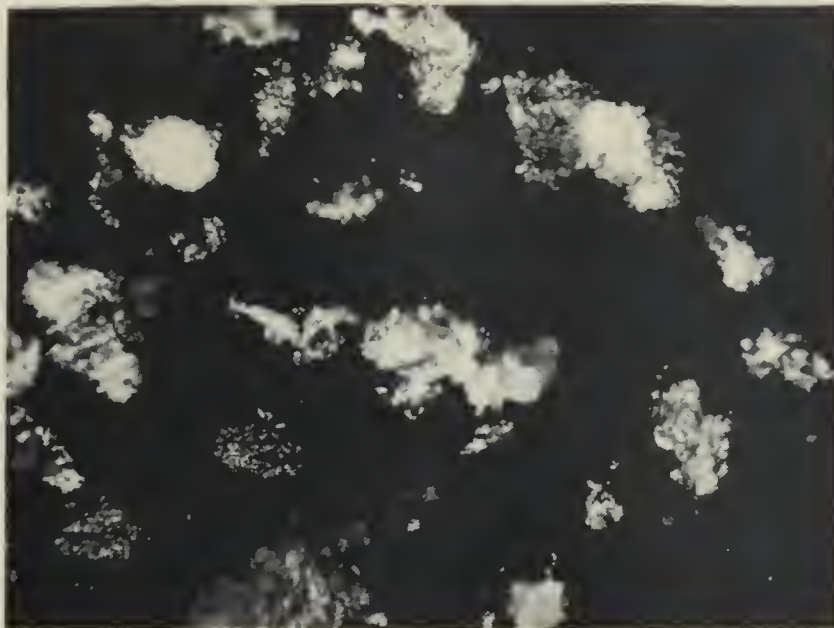


Figure 12.- Gold from Elkora Mines tailing. (Magnification 120 X.)

These gold particles were not recovered by flotation because of the existence of a shell or coating of oxidized iron. The surface contamination was sufficiently magnetic to permit segregation by a magnetic point. Identification as metallic gold was possible only after destroying the coating with $\text{HCl} + \text{SnCl}_2$.

Most of the coated gold particles are close to the 200-mesh size range, but some larger particles are shown in the picture with them. Note that all particles have very irregular surfaces from which coatings are difficult to remove.

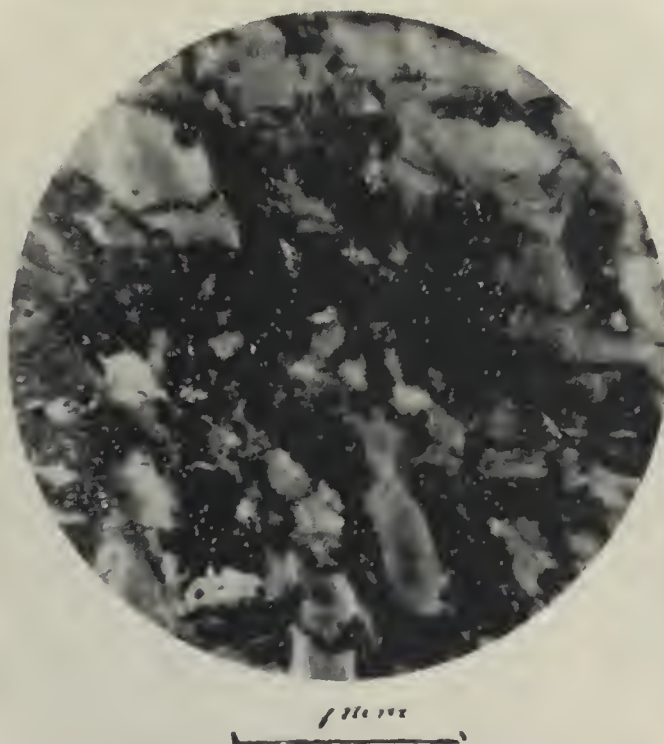
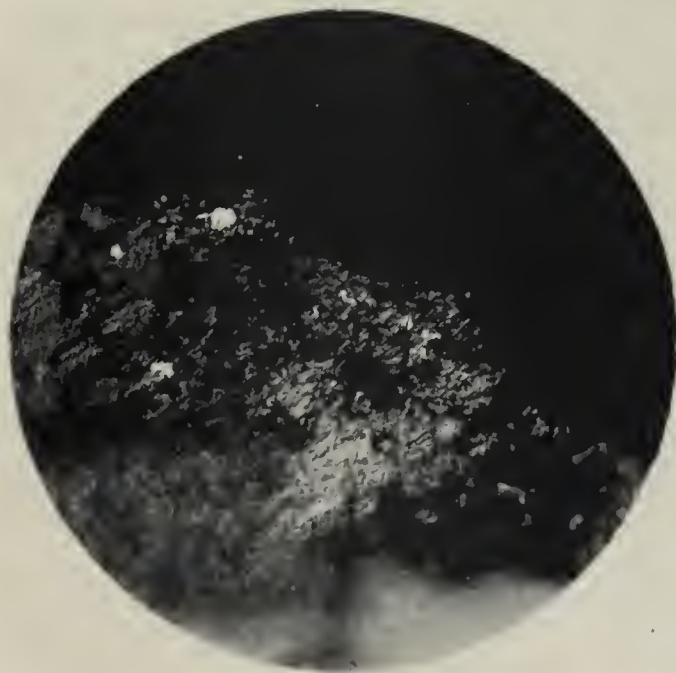


Figure 13.- Tintic standard lead sulphate.
(Magnification 25 X.)

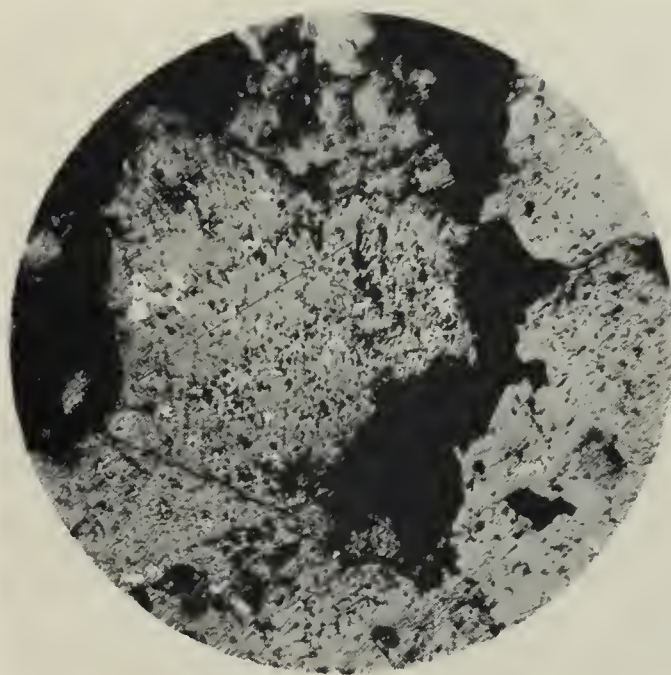
Shows intimate association of the dark- and light-colored varieties. Assays made on picked light and dark material show that the dark sulphate is invariably much richer in silver.



. 2 mm

Figure 14.- Tintic standard lead sulphate.
(Magnification 150 X.)

Shows association of metallic silver in very small particles. The largest particle shown is of approximately 800-mesh diameter and the smallest particle is less than 1,600-mesh.



. 2 mm

Figure 15.- Metallic silver in Tintic Standard lead carbonate. (Magnification 150 X.)

Large black areas are holes. White areas are metallic silver. Gray areas are lead carbonate. The largest silver particles in the field are 0.03 mm, or 30 microns in diameter (smaller than 400-mesh). The smallest particles visible are approximately 1 micron, or the equivalent of the openings in a theoretical 9,200-mesh sieve. Because both lead carbonate and lead sulphate contain metallic silver particles of too fine a size to be freed by grinding, any loss of these two minerals in tailings entails a possible loss of silver.

Metallizing by Long Grinding followed by Flotation

An interesting development in the treatment of refractory ores has resulted from the observation that when the siliceous Eureka Lilly gold ore was ground for a long time with iron balls, the gold became smeared with abraided iron and could be removed with a magnet. When siliceous lead-silver ores were ground for long periods under similar conditions, it was learned that much of the lead was metallized and could be recovered subsequently by panning or floating. Although lead carbonate and lead sulphate are only slightly soluble, on long-continued grinding the small amount of lead in solution is precipitated instantly, so that the actions of dissolving and metallizing the lead proceed toward completion as one of the products of the reaction is constantly removed.

This new angle of attack was explored by trying synthetic mixtures of various lead minerals picked from samples of the Tintic Standard ore. It was learned that these and other oxidized minerals were metallized, to some extent at least, by long grinding and that the silver content of such minerals often was more metallized than the lead. Thus, in parallel tests some 93 percent of both the silver and lead in silver-bearing anglesite, 47 and 78 percent, respectively, of the lead and silver in cerussite, and 48 and 31 percent of the lead and silver in plumbojarosite were metallized. Silver precipitation from plumbojarosite is assisted by the presence of lead. It was found, also, that the addition of anglesite to a gold ore assisted in the recovery of the gold after fine grinding. In general, the carbonates metallize more slowly than do the sulphates. It was found that cerargyrite and silver-bearing wulfenite also were fairly amenable to this treatment. In these tests the minerals and quartz were ground together at room temperature in aqueous pulps for 8 hours.

Work is now in progress to determine the factors governing the rates of metallizing and of recovering lead and other minerals in oxidized ores and of separating such metallized portions or valuable fractions of the ore as concentrates. For ores, it has been learned that it is necessary to have considerable sharp, hard gangue present to keep the balls clean so that the lead and other metals may be precipitated effectively. Soft ocherous gangue and the soft material in leaching-plant residue interfere. On trying to increase the surface of the iron by using shot instead of balls it was learned that the grinding media must be heavy enough so that they rub on each other, otherwise the precipitated metal is not cleaned completely from the balls. It is also necessary to run the mills at such speed that a true grinding action takes place; if the mill be run at high speed, so that the balls centrifuge, the lead is not removed effectively from the ore. With clean ball surfaces, the rate of metallizing was not increased by adding a small amount of fine zinc powder.

The rates of dissolution, metallization, and recovery are other factors requiring investigation. Expensive solvents or those that are consumed rapidly, as is strong sulphuric acid by carbonates, cannot be used in plants to hasten these reactions. Brine is one effective, cheap reagent that does

not react prohibitively with the gangue. Sulphuric acid also is effective in some instances, but, as stated, this cannot be used with some types of ore. Since rates of dissolving are accelerated by an increase in temperature, it has been found advisable to use as hot solutions as may be feasible. Heating costs are not excessive as only 0.6 pound of solution is used per pound of ore.

It has been found necessary to try various types of flotation reagents. The most effective procedure to date has been to float all of the valuable minerals possible after a preliminary grind period of about half an hour. The reagents used after this preliminary grinding period will vary, of course, with the type of ore used. In many cases, pentasol xanthate plus B-23 and other frothers have been found effective. It is advantageous, in the preliminary flotation test to sulphidize many ores to remove as much of the valuable minerals as possible in the first step. The commonly effective reagent combination for the second stage after the long grinding period was an emulsion of X-1 with dixanthogen. Either the amyl or the ethyl compound can be used.

To summarize what has been accomplished in the development of this process: The total grinding period has been cut to 1-1/2 hours - 1/2 hour for the normal preliminary grinding step and 1 hour for the "long grind." It has been found possible to reduce the Tintic Standard tailing from about 6.0 to 1.85 ounces of silver and to under 1 percent of lead while floating 93.4 percent of silver and 82.2 percent of the lead. The concentrates are of good grade. Up to 94-1/2 percent of the silver has been floated, of which about 83 percent was floated in the first stage under favorable conditions. The other well-known oxidized ore of the district, the Chief Consolidated, contains much more lime and oxidized lead than does that from the Tintic Standard. Over 90 percent of the silver has been floated from the Chief Consolidated ore and a tailing made containing less than 1.5 ounces of silver per ton. About 83 percent of the lead is floated in a product of marketable grade. Using current smelter rates, a possible return is indicated of \$3 more per ton than is received at present for one of these ores.

Pyrometallurgical Work on Lead

Work on the direct metallizing of lead from rich galena concentrate has been recessed pending the receipt of funds necessary to carry on this investigation. A roaster is now being built in which it will be attempted to sinter richer lead concentrate than has been possible heretofore.

Removal of Bismuth from Lead Products

From Metallic Lead

Little success has been attained in this investigation. Nickel is reported to form intermetallic compounds with bismuth and to be practically insoluble in lead at temperatures near the melting point of the latter, and,

therefore, might be expected to act as a debismuthizing agent. After considerable experimentation it was found that the difficulty in disseminating nickel through lead, due to the high melting point of nickel, could be avoided by adding nickel as the chloride. Unfortunately, it was found that the nickel was not insoluble in lead at temperatures near its melting point (327°C.), so that any bismuth that may have combined with the nickel was left in the lead. The same lack of success was found on trying to use manganese and also carbon silicide and silicon bisulphide for the removal of bismuth from lead.

From Galena Concentrates and Residues

If bismuth could be removed from products before sending them to the lead blast furnace the necessity for debismuthization would be eliminated. Bismuth is likely to be present in an ore in several mineralogical forms, and no apparent success has been obtained to date in its complete removal by flotation.

On leaching a raw flotation concentrate with ferric chloride in an acidic brine, it was found possible to dissolve 41 to 50 percent of the bismuth and only 9 to 11 percent of the lead. (This lead product contained 0.028 percent of bismuth.) If the same relative rate of dissolution could be maintained, one could dissolve over 90 percent of the bismuth and less than 20 percent of the lead; it would then be possible to precipitate the dissolved bismuth on lead shot. Unfortunately, when it is attempted to prolong the time of treatment, the rate of dissolving bismuth decreases and that of lead increases, so that by the time 69 percent of the bismuth is dissolved, 23 percent of the lead is dissolved, also. On working with a product containing over 0.1 percent of bismuth, where the bismuth was associated with copper instead of lead, it was found that still less of the bismuth was dissolved in acidic brine.

A few tests were made to remove bismuth by leaching roasted products. Ferric chloride dissolved more lead than bismuth from roasted products in which the bismuth was supposed to be associated with copper. Sulphuric acid-brine solutions containing small amounts of ferric chloride dissolved several times as much bismuth as lead. A lead calcine containing 0.032 percent of bismuth and 35 percent of lead, in which nearly all the residual sulphur was as sulphate, was leached with 15 percent sulphuric acid solution. About 30 percent of its bismuth was dissolved with only about 2 percent of its contained lead. The silver present was found to complicate things still more. On sulphating with sulphuric acid and releaching with a 10 percent solution, about 30 percent of the bismuth and 20 percent of the silver were dissolved. On increasing the temperature of roasting to 600°C. , 75 percent of the silver was found to be soluble in sulphuric acid but the percentage of bismuth dissolved dropped to 18. On fuming the roasted calcine with sulphuric acid and then leaching with water, about 65 percent of the silver and 18 percent of the bismuth dissolved. Adding a small amount of ferric sulphate did not help matters.

A leached calcine residue containing 355 ounces of silver, 0.21 percent of bismuth, and a small amount of lead was procured from a smelter. This product was leached with acidic solutions of various strength. Solutions containing 25 to 50 percent acid gave about the same maximum results, extracting about 85 percent of the silver and 55 percent of bismuth. After sulphating and leaching with water, none of the bismuth and 68 percent of the silver was dissolved. The silver in the product probably could then be leached out fairly successfully by repeating the operation and leaving the bismuth behind. In general less bismuth than silver is made soluble by sulphating roasting.

Publications

<u>Title</u>	<u>Authors</u>	<u>Where published</u>
Smelting in the Lead Blast Furnace	G. L. Oldright and Virgil Miller	Lead and Zinc, vol. 121, 1936, American Institute of Mining and Metallurgical Engineers, pp. 82-105.
Drosses in Lead Smelters	C. M. Dice, G. L. Oldright, T. B. Brighton	Lead and Zinc, vol. 121, 1936, American Institute of Mining and Metallurgical Engineers, pp. 127-159.

PRECIOUS-METALS SECTION

Metallurgy of Gold

Gold Loss in Dredging

The report for 1936 (R. I. 3331) gave the findings of the preliminary investigation on the loss of gold in dredging. The work was continued, and a progress report^{8/} was made to the American Institute of Mining and Metallurgical Engineers.

Sampling to show the true value of waste dredge products has not proved to be practical and the laboratory tests do not compare well with dredging operations. However, our research covering a multiple of experiments on many dredge samples does indicate a considerable percentage of fine gold that is not recovered by the usual riffle amalgamation method, and we have demonstrated that the slow-curing oil residuum is a good collector of this fine gold.

Each dredging field need be considered as a separate problem because the conditions vary with the locality. Consultation with staff and operators on each of the dredges visited in considering losses that are based upon fire assays and microscopic examination has formed the basis of our informal cooperation.

Our activities must necessarily be confined largely to a discussion of losses and additional means of improving recovery because of the cost of experiments on a practical scale. Of first importance is the conviction that actual losses represent possible profit, which creates an incentive for large-scale research. A better general idea of the gold loss plus the increased price of gold has resulted in considerable improved recovery by newer designs in each of the various dredging fields.

Diaphragm jigs have replaced the usual riffles on two of the larger dredges and have extended to the smaller dragline type of excavator. On one large dredge, recoveries have been improved by classification, by means of which the fine material is passed over separate riffles more slowly, and the rougher concentrates are retreated by centrifugal concentration to obtain a product that is reground in a tube mill before passing it on to the special amalgamation riffles.

Our study is largely concerned with the development of methods to recover some of the extremely fine gold that is now lost in the improved dredging operations. The most favorable laboratory results were obtained with a mixture of 10 percent of crushed charcoal and 90 percent warm residuum. It is an excellent collector of gold, readily separates from the

^{8/} Leaver, E. S., and Woolf, J. A., Gold Dredging in California and Methods Devised to Increase Recovery: Am. Inst. Min. and Met. Eng., Tech. Pub. 792, 1937, 18 pr.

agitated pulp, and quickly forms on the surface of the water. By diluting the collector with kerosene, the gold and charcoal settle readily, so that it is practical to pour off the residuum-kerosene mixture, which can be distilled for the reuse of both products. The gold-charcoal residue may be burned and the resultant ash melted for the recovery of the gold as bullion.

Amalgamation

In the usual amalgamation process, the clean-up consists of more or less fouled mercury and amalgam. It is compressed within chamois or a special canvas filter to separate the mercury clean enough for its reuse. This method will remove the sulphides of arsenic or antimony as well as amalgams of antimony, or copper, and most other impurities. It does not remove all of a zinc-mercury amalgam, but this is not encountered in milling ores.

Vibrators may aid amalgamation when the surface of the mercury trap is kept sufficiently mobile to break up or prevent packing. Compressed-air vibrators are favored because of better control. However, vibrators have not been considered of enough importance to warrant their regular installation.

Gold in Pyrite or Base-Metal Ores

Considerable work has been done on this problem during the last few years. Simple pyritic ores, in which the gold is associated largely with the pyrite, usually can be treated by one of the following methods or a combination of them:

1. Flotation and smelting of the concentrate.
2. Flotation, roasting of concentrate, followed by cyanidation of calcine.
3. Roasting of original ore followed by cyanidation of calcine.
4. Direct cyanidation of original ore.
5. Flotation with some treatment for the concentrate and direct cyanidation of the flotation tailing.

In the case of more complex pyritic gold ores that contain copper, lead, arsenic, or antimony, the problem of effecting an economic recovery of the gold may be exceedingly difficult. In such ores part of the gold may be associated with the pyrite, the remainder being distributed unevenly with the other base-metal sulphides and nonmetallic gangue minerals. Consequently, floating only one or even all of the sulphides may not effect a high recovery of the gold. Furthermore, the concentrate produced in such cases is likely to be too low grade for direct smelting and often is not amenable to roasting and cyanidation because of its complex character.

The presence of arsenic or antimony sulphides in a pyritic gold ore is decidedly detrimental because of their solubility in alkaline cyanide solution. If such an ore or concentrate is roasted before cyanidation, gold may be locked up during the roast by the formation of ferrites, so that cyanidation of the calcine results in low extraction of the gold.

Recently, a cooperative agreement was entered into between a mining company in Nevada and the Bureau of Mines, whereby intensive work is being done on a pyritic gold ore containing arsenic sulphides. This company has developed a large tonnage of such ore and plans active milling operations as soon as a suitable method of treatment is worked out. This ore presents a difficult treatment problem, but results to date are encouraging enough to indicate that an economic scheme of treatment for this ore will be devised soon.

Flotation of Silver Minerals

Many natural ores, in which gold is the principal valuable constituent, contain enough silver to warrant thorough investigation for the best method of its recovery. Generally speaking, gold is extracted by cyanidation more easily than is silver. However, in the case of many complex gold ores containing minor amounts of silver, flotation has been successful in increasing the recovery of both gold and silver. Lead and copper ores that contain some gold and silver usually are treated by flotation, and it is desirable to recover as much of the precious metals as possible by this method. Therefore, the successful application of flotation to the recovery of silver minerals and of silver associated with base-metal minerals would serve to increase materially the amount of silver recovered in many milling operations. In addition, there are small properties with tonnages too low to warrant the installation of a cyanide plant but which might be operated successfully by flotation. These reasons prompted the undertaking of the flotation of silver minerals problem by Bureau of Mines.

Most of the work in connection with this problem has been done with "synthetic ores" composed of a pure silver mineral and sea sand. In general, all of the common silver minerals - argentite, cerargyrite, polybasite, proustite, pyrargyrite, and stephanite - are floated readily from such synthetic ores by the use of xanthate, aerofloat, and cresylic acid. Finely divided metallic silver floats well when not tarnished or contaminated. The addition of a considerable percentage of talcose or iron oxide slime to the synthetic ore usually does no harm except to lower the grade of concentrate, and this effect can be overcome partly by the careful use of starch in the flotation circuit.

The addition of lime to the flotation circuit during the flotation of silver minerals may produce no special effect other than to change the grade of concentrate. As a rule, lime tends to decrease the grade of the concentrate when floating silver minerals from synthetic ores; in any case, when silver minerals are to be floated, the addition of lime to the flotation circuit should be investigated thoroughly before it is applied in practice.

Sodium sulphide is an active depressant for all of the silver minerals when floating synthetic ores. With some natural silver ores, sodium sulphide may be used with no depressing effect on the silver, but no ore has been tested for which the addition of sodium sulphide was actually beneficial if

the silver occurred as a distinct silver mineral and not associated with oxidized base metal minerals. When it is planned to use sodium sulphide to aid the flotation of base-metal minerals, thorough study of its effect on any silver minerals present should be made.

As to the flotation of silver minerals in general, they are floated readily from a quartz gangue if they are reasonably pure and not too badly tarnished or contaminated with slime. Xanthate, aerofloat, and cresylic acid are effective reagents for the flotation of silver minerals. The effect of addition agents, such as starch, sodium sulphide, and lime, should be studied carefully before adding them to the flotation circuit when treating an ore containing silver minerals.

Flotation of Scheelite

Scheelite is the natural mineral that supplies almost all of the tungsten produced in this country. In crushing scheelite ores for milling, the scheelite slimes readily, which accounts for a large percentage of the tungsten lost in the usual gravity concentration.

Our laboratory flotation tests for the recovery of scheelite from waste slime produced during gravity concentration of scheelite ore were so satisfactory that under a cooperative agreement with Bureau of Mines, the Nevada Massachusetts Co., Mill City, Nev., installed a 75-ton-per-day pilot flotation plant to apply the process.

The present slimed portion of the gravity tails constitutes 25 to 35 percent of the mill tonnage and contains 0.25 to 0.50 percent WO_3 . The slime formerly wasted, which constitutes the flotation feed, contains about 5 percent of sulphides that would float during the scheelite flotation. Therefore, a prior sulphide float is made before the scheelite to eliminate the major portion of such sulphides and cause less contamination of the scheelite rougher concentrate.

It has not been proved practical to produce a marketable product by direction flotation, as to do so would be a sacrifice shown in much lower recovery. The purpose of our flotation plan is to separate the extremely fine scheelite from its slime and thus obtain approximately 10 percent WO_3 concentrate, which is subsequently raised to a 55 to 65 percent product by a gravity concentration on the Diester slime table.

Figure 16 is the flow sheet of the flotation plant:

Control of the pulp alkalinity within narrow limits (pH 8.0 to 8.5) is essential for best results, such being accomplished by soda ash, which also serves as a water softener. It is important that the water used in the flotation circuit be fairly free from hardness or that it be softened prior to the introduction of reagents for flotation of scheelite.

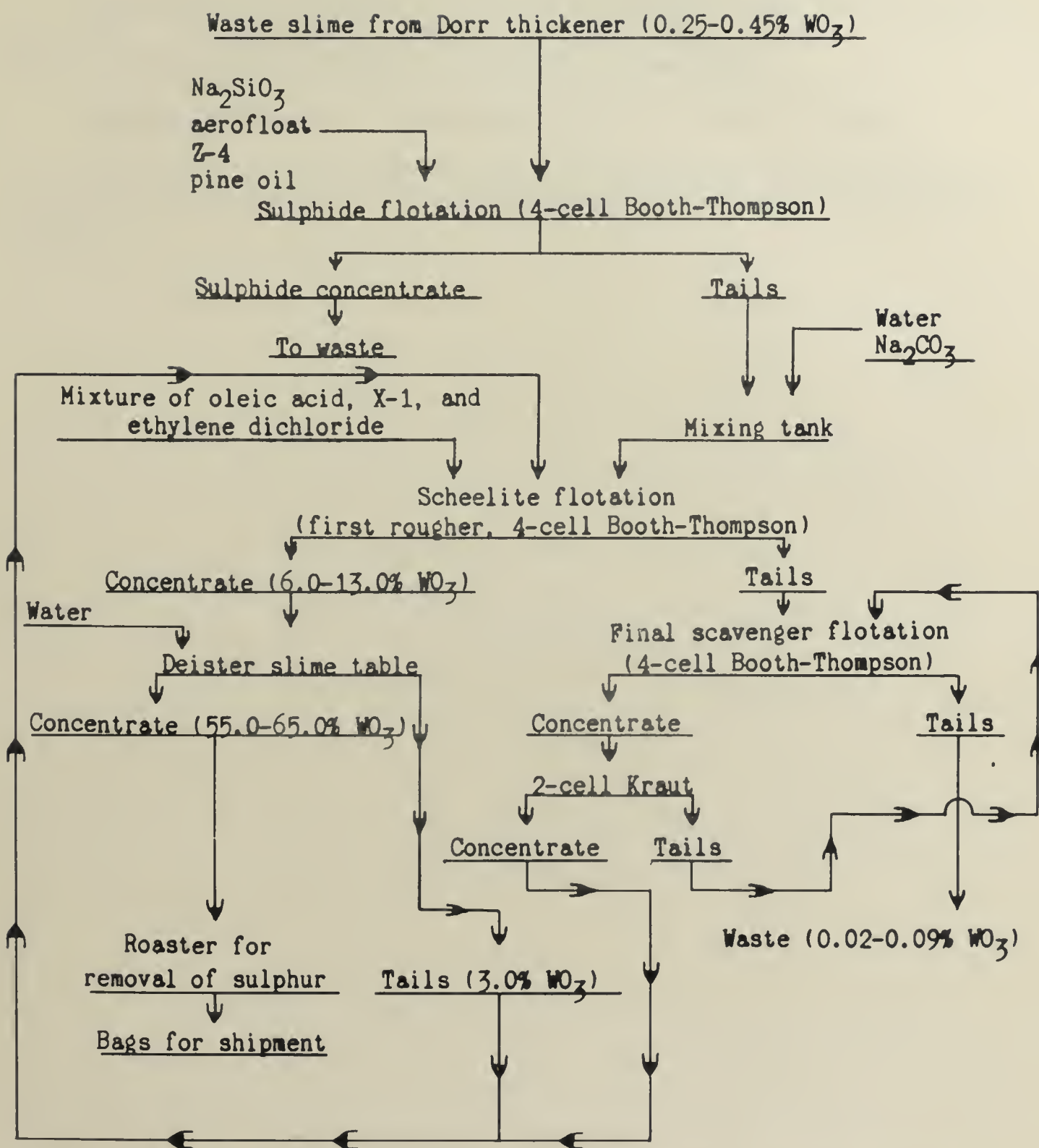


Figure 16.- Flow sheet of flotation section in Nevada-Massachusetts mill, Nevada.

Approximately equal parts of oleic acid, Emulsol X-1, and ethylene dichloride for the solvent are satisfactory for scheelite flotation.

Extractions of scheelite from slime by flotation have been as high as 91 percent. Marketable concentrates up to 65 percent WO_3 have been obtained from the slime table treating the flotation concentrate.

Typical results of the various mill runs are shown in Table 1.

Results to date in the 75-ton plant show definite possibilities for future recovery of scheelite by flotation and this one to be a profitable venture.

TABLE 1. -- Flotation of scheelite from waste slimes at Nevada-Massachusetts new plant

Date, 1937	Flotation, percent WO_3			% re- covery	Table concen- tration, per cent WO_3		Reagents, pounds per ton dry slime								Turk- ey red oil
	Heads	Tails	Concen- trates		Concen- trates	Tails	Na_2CO_3	Na_2SiO_3	Caleic acid	X-1	Ethylene dichloride	B-22	Alpha- sol		
4/23	0.36	0.09	6.25	75.00	56.74	2.53	9.5	0.84	0.97	1.16	0.96	-	-	-	
4/27	.34	.04	9.73	88.25	65.92	5.43	9.5	.84	.97	1.16	.96	-	-	-	
5/ 5	.24	.02	6.95	91.60	51.48	3.56	9.5	.84	.97	1.16	.96	-	-	-	
5/10	.27	.05	5.10	81.42	52.36	2.56	9.5	.84	.97	1.16	.96	-	-	-	
5/14	.34	.07	12.57	79.40	39.00	2.81	9.5	.84	.97	1.16	.96	-	-	-	
5/17	.47	.13	9.70	61.65	64.10	2.17	8.5	1.14	.16	.16	.16	0.04	0.16	-	
5/20	.41	.14	8.55	65.82	42.70	2.93	8.5	1.14	.16	.16	.16	.04	.16	-	
5/24	.41	.07	5.90	82.91	47.20	4.10	6.6	1.14	.42	.26	.26	.12	-	0.38	

COPPER METALLURGY SECTION

During the past year the copper metallurgy section has devoted its energies to the study of two problems - the recovery of sulphur from smelter smoke and factors governing the flotation of oxidized minerals of copper. The second problem is new this year, while the former is in its second year and during the latter part of 1936 country-wide observations were made of smelters in operation.

Recovery of Sulphur from Smelter Smoke

Work on this problem was devoted, for the most part, to a study of possible methods of concentrating the sulphur dioxide (SO_2) content of a dilute waste gas into an enriched fraction that would form suitable raw material for the production of elemental sulphur by reduction.

Further experimentation was directed to the development of an absorption process that would function satisfactorily at higher temperatures. An investigation into the Hall process of roasting sulphide concentrates to give an oxide calcine and elemental sulphur furnished data showing the process to be rather unsatisfactory. The use of aqueous solutions of substituted ammonia and ammonium sulphite as thermally reversible solvents for SO_2 has been investigated further.

Although the use of aqueous solutions of amines as thermally reversible solvents for weakly acidic gases has been patented, there is virtually no information available concerning the behavior of such solutions when used as solvents for SO_2 , and previous qualitative tests of such material were so favorable that data of a more quantitative nature seemed desirable. Accordingly, work was started on the measurement of the partial pressure of SO_2 gas in equilibrium, at various temperatures, with amine-water mixtures of different proportions which were partly saturated with the gas. A study of the density of the same solutions also is being made.

A dynamic method for the vapor-pressure measurements was used, in which a measured volume of nitrogen was passed in contact with the amine solution and then the amount of SO_2 in the nitrogen was determined.

The solutions whose properties were to be studied were made up from samples as supplied for industrial use. These are not pure single substances but a mixture of several amines, one of which greatly predominates. The values determined are, therefore, not fundamental for the properties of a single pure substance but do give a fair idea of the limits of application of such solutions as far as temperature and absorption capacity are concerned. The data for equilibrium partial pressures are given in tables 2, 3, and 4. The values for actual SO_2 content and density of the solutions are not completed, so at present the data allow estimating how completely SO_2 may be removed from a given mixture but not what volume of solvent will be required per volume of gas.

TABLE 2. - Partial pressure of SO₂ over amine-water solutions at 50° C. containing 30 percent amine and 75 percent saturated with SO₂

Amine	Partial pressure of SO ₂ , mm Hg
Monoethanol.....	1.6
Diethanol.....	4.0
Triethanol.....	34.0
Ethylene diamine.....	2.3
Diethylene triamine.....	.4
Triethylene tetramine.....	10.9

TABLE 3. - Partial pressure of SO₂ over solutions of SO₂ in triethylene tetramine-water mixtures

Amine concentration, percent by volume	Percent saturation with SO ₂	Partial pressure of SO ₂ , mm Hg at -			
		35°C.	50°C.	75°C.	90°C.
10	50	---	---	0.06	1.30
20	50	---	---	---	.20
30	50	---	---	---	.07
10	60	0.4	1.02	4.40	10.40
20	60	---	---	2.90	6.20
30	60	---	.16	2.50	5.70
10	75	32.0	67.00	297.00	---
20	75	5.4	16.00	72.00	156.00
30	75	4.6	11.00	60.00	118.00
10	90	382.0	---	---	---
20	90	320.0	---	---	---
30	90	199.0	302.00	---	---

TABLE 4. - Partial pressure of SO₂ over solutions of SO₂ in diethylene triamine-water mixtures

Amine concentration, percent by volume	Percent saturation with SO ₂	Partial pressure of SO ₂ , mm Hg at -			
		35°C.	50°C.	75°C.	90°C.
10	50	--	0.80	2.90	7.0
20	50	--	--	--	.8
30	50	--	--	.17	.5
40	50	--	--	--	.6
10	60	0.16	1.90	12.00	39.0
20	60	--	.70	6.70	16.0
30	60	--	.60	3.60	9.0
40	60	--	.60	2.40	7.0
10	75	3.50	89.00	315.00	--
20	75	1.60	9.00	51.00	129.0
30	75	.10	.40	30.00	93.0
40	75	--	.04	22.00	59.0
10	90	307.00	--	--	--
20	90	59.00	231.00	--	--
30	90	13.00	167.00	230.00	--
40	90	3.2	21.00	174.00	--

Publications

The work on sulphur recovery has been reported in Bureau of Mines Report of Investigations 3339, Fixation of Sulphur from Smelter Smoke, by R. S. Dean and others, 1937, 51 pp., and a thesis by J. F. Michaelson in the University of Arizona Library intitled, "Experimental Work in the Recovery of Elemental Sulphur by the Hall Process."

Inhibition of Oxidation of Ammonium Sulphite Solution

Further experiments in this work, first reported last year, have shown that para aminol phenol or para phenylene diamine will reduce the rate of oxidation of ammonium bisulphite solutions to one-fifth of the normal value when used to the extent of 1 part inhibitor to 20,000 parts of solution by weight. The protection thus conferred against oxidation is effective for several hours at room temperature but is quickly destroyed if the solution be heated to 50°C. or over.

Flotation of Oxidized Copper Minerals

This work has mostly consisted of tests with a 50- or 125-gram machine on synthetic charges composed of a natural pegmatite containing quartz and slightly weathered orthoclase, to which was added enough pure chrysocolla to make the copper content of the charge about 0.6 percent. Both the pegmatite gangue and the chrysocolla were ground to pass 100 mesh.

Of a considerable number of reagents tried, only the alkali soaps of the fatty acids and the high xanthates acted as collectors for chrysocolla, and most of the work was done with the fatty acids. The factors that influence the flotation of chrysocolla by this type of reagent may be outlined briefly as follows:

The pulp must be relatively free from soluble alkaline earth or heavy metal ions, because these react with the soap added as a reagent to form insoluble soaps. The latter, although they are, to some extent at least, collectors for malachite and sulphides, are quite useless for chrysocolla, so that the presence of such soluble salts in the water or ore increases the reagent requirement.

The pH of the pulp exerts a marked influence, largely because it determines the form in which the reagent is present. In a pulp of low pH, the active agent is probably the free fatty acid, regardless of whether the free acid or the alkali soap was added originally. Likewise, in a pulp of high pH, the alkali soap is present. The free fatty acid does not show much selectivity as a collector for chrysocolla. If enough reagent is used to give good recovery, the concentration ratio will be low, probably less than unity. With a pH of 8 or 9, fairly complete recovery and a concentration ratio varying from 5 to 8 may be obtained. Reagent consumption will depend largely on the nature and amount of slime present. With carefully deslimed material it may be as low as 3 pounds per ton of charge, and with certain types of slime as high as 10 or 12 pounds per ton.

In floating chrysocolla, best results are obtained by using soap as the only reagent. Enough must be used to give a froth before the chrysocolla will float, and nothing is gained by trying to use other frothers, such as pine oil, X-1, or the alkali sulphates. Dispersing agents, such as metaphosphates and soluble silicates, are definitely harmful and, of course, alkali carbonates or hydroxides must supplant lime in regulating the pH. In general, soaps from animal fatty acids are much more effective than those of vegetable origin.

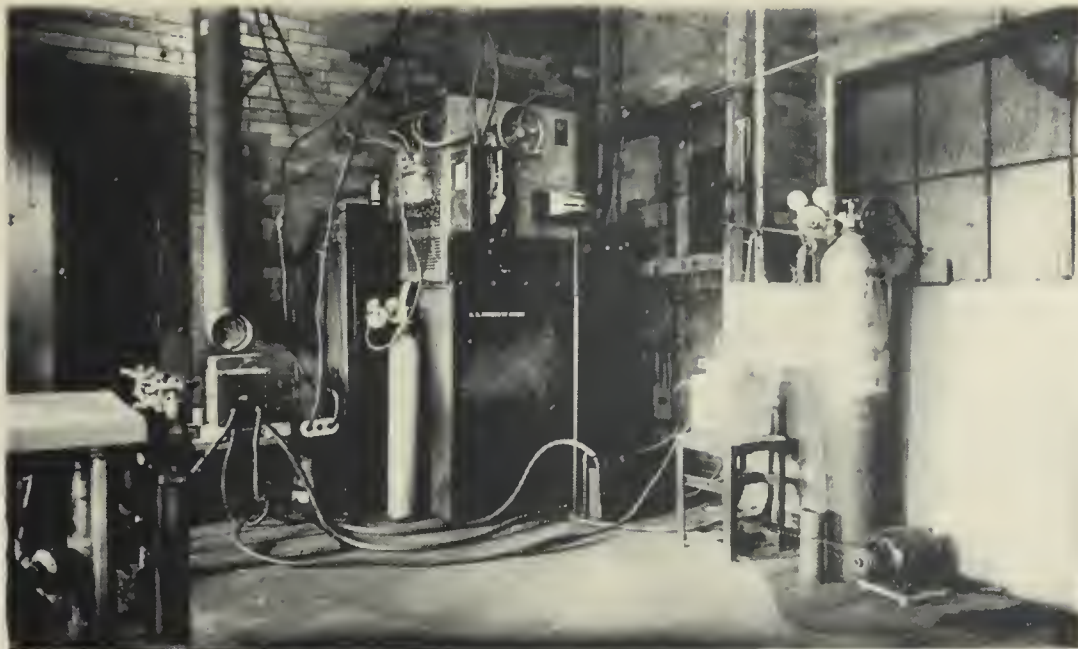


Figure 17.—High-frequency induction furnace laboratory.

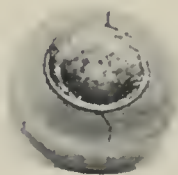


A. TOP VIEW



B. SECTION THROUGH CRUCIBLE AND CHARGE

Figure 18.—Relative position of FeS and slag, cooled while rotating.



A. TOP VIEW



B. SIDE VIEW, SLAG REMOVED

Figure 19.—Relative position of FeS and slag; rotation stopped; crucible quenched.

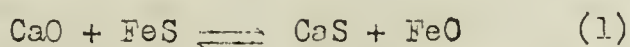
BLAST-FURNACE STUDIES SECTION

Desulphurization Studies

Mechanism of desulphurization

The work of the Blast-Furnace Studies Section has been directed toward a more critical study of the mechanism of desulphurization of metal by slags. The manufacturer of steel has been forced to meet more exacting specifications, due to the more rigid requirements demanded of steel by the consumer. It is desirable to remove as much as possible of the sulphur in the blast furnace.

The first step in desulphurization is accepted as being the result, chiefly, of the reaction



Other bases, such as MnO and MgO, may react to a lesser degree.

In the presence of carbon, another reaction plays a significant part,



If reaction (2) were allowed to proceed to equilibrium, the FeO would approach zero and desulphurization would be substantially complete. Such a condition is never reached in the blast furnace because FeO is being replaced continuously in the form of incompletely reduced iron ore and as a result of oxidation in the combustion zone.

To determine the effect of FeO on desulphurization, a study of equilibrium in reaction (1) was started. This necessitated the elimination of graphite as a refractory material for holding slag and metal. After unsuccessful trials with refractory mixtures, a furnace was constructed to utilize a rotating liquid crucible composed of one of the reacting substances. This type of furnace, shown in figure 17, has been employed by other investigators, but as each individual application must meet different requirements, a number of changes was necessary before satisfactory operation could be obtained. Figure 18 shows a melt that was allowed to solidify while in rotation, while in figure 19 is seen a melt after quenching for analysis.

Preliminary values have been obtained for equilibrium at 1,600° C., using purified FeS as the liquid crucible to hold the standard calcium-aluminum-silicate slag. Values will be obtained for other temperatures and slag compositions. Before the equilibrium constants can be evaluated, however, it will be necessary to know the amount of molecular FeS present in the slag at different temperatures or prove that it is substantially insoluble in the slag. At the present time no method is available for determining the amount of FeS or MnS that may exist in blast-furnace slags while desulphurization is taking place.

A study of the absorption spectra of slags containing iron and sulphur has been started and may develop into a method for determining FeS in slags. When either FeS or FeO was added to a slag containing CaS a highly-colored brown to black slag resulted. Similarly colored slags were obtained when the standard calcium-aluminum-silicate slag was allowed to react with molten ferrous sulphide. It is thought that the intensity of the color is proportional to the amount of ferrous sulphide present.

Relative desulphurizing power of slags high in MgO.

In the previous annual report reference was made to unpublished data on the desulphurizing power of slags containing up to 20 percent MgO. These values will be included in a manuscript that is being prepared for publication.

In general, the same may be said regarding the desulphurizing power of slags containing 15 and 20 percent MgO, as was reported for the 5 and 10 percent MgO slags. When used to increase the basicity and thin out the slag, MgO was an aid to desulphurization but was less effective than CaO at the same basicity.

Oxides in Basic Pig Iron and in Basic Open-Hearth Steel

In the basic open-hearth process for making steel, open-hearth men long have attributed difficulties in making satisfactory steel to characteristics of the pig iron not disclosed by ordinary analysis. An investigation of the extent to which oxides in pig iron affect the oxides or nonmetallic inclusions in steel was started last fiscal year in cooperation with the Inland Steel Co. and completed during the present year.

The amount and composition of the oxides were determined in several hundred casts of basic iron produced in a 700-ton furnace. No relation was found to exist between the oxides in the pig iron and in the steel made from it, as shown in figure 20. The total oxides in the iron varied from 0.012 to 0.313 percent. As the total oxides in the pig iron increased, the percentage of FeO in the oxide residue increased. The oxide residue from 194 casts averaged 22 percent SiO₂, 14 percent MnO, and 64 percent FeO. Oxides determined on a considerable tonnage of iron held in the mixer before transfer to the open hearth indicated a loss of oxides between the blast furnace and open hearth as shown in table 5.

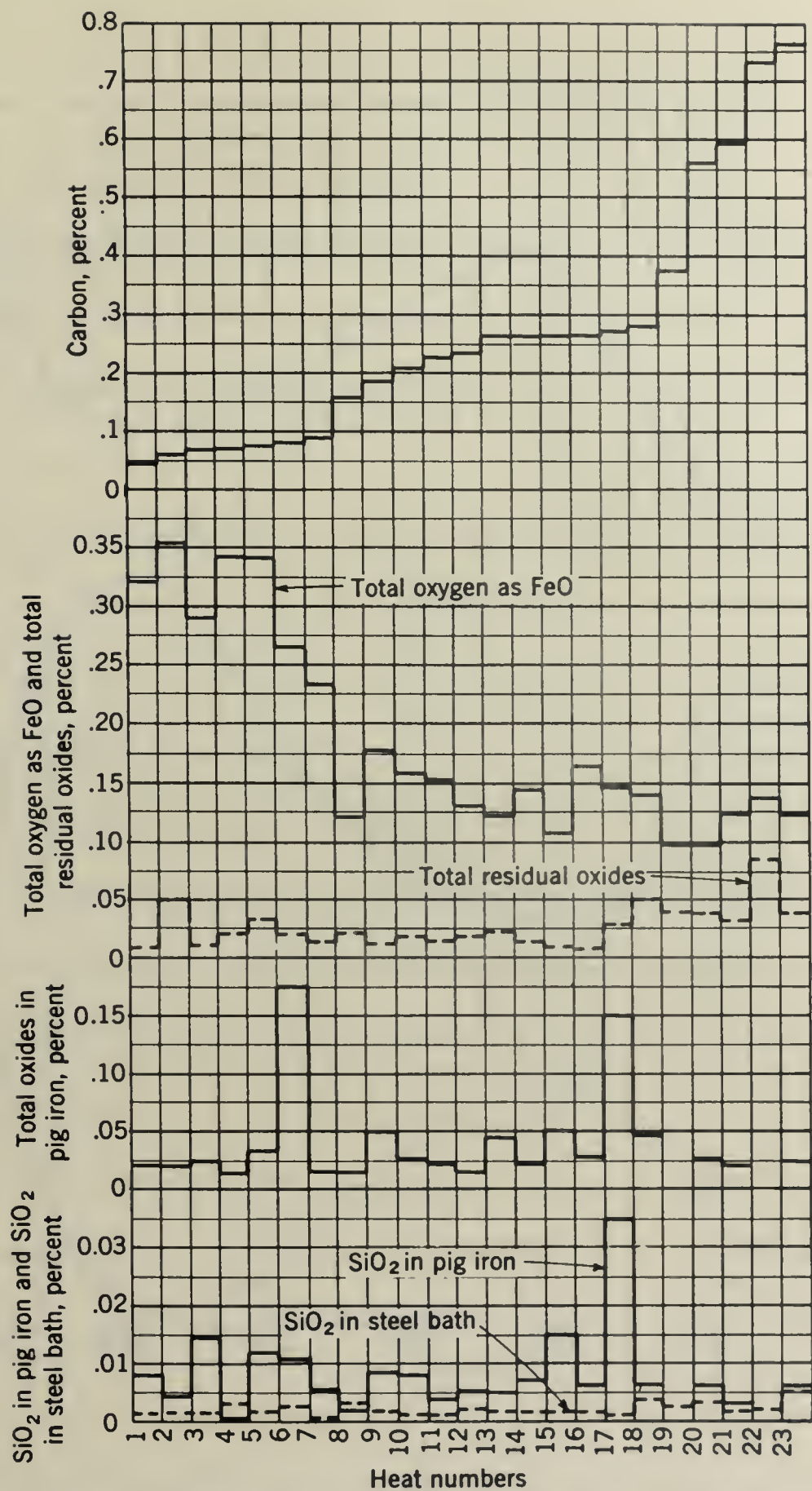


Figure 20.—Oxides in basic pig iron and open-hearth steel.

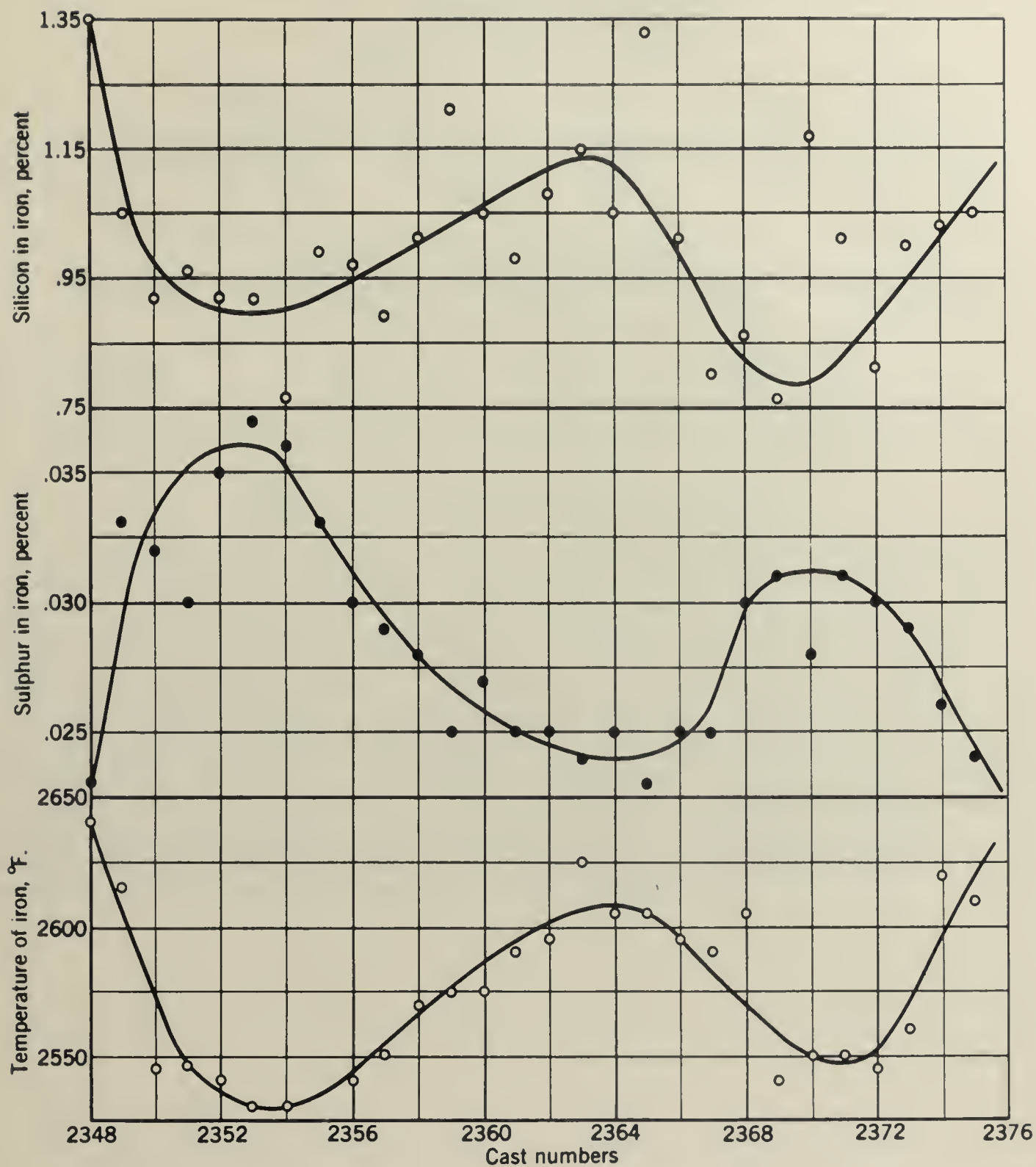


Figure 21.—Relation between silicon and sulphur and the temperature of consecutive casts.

TABLE 5. - Oxides in 194 casts at blast furnace and iron used in 23 heats.

	Oxides in sample, percent				Total oxides
	SiO ₂	MnO	FeO	Al ₂ O ₃	
Blast furnace	0.011	0.007	0.032	---	0.050
Open hearth	.0076	.0036	.0245	0.0005	.0362

Composition of residue, percent				
	SiO ₂	MnO	FeO	Al ₂ O ₃
Blast furnace	22.0	14.0	64.0	---
Open hearth	21.0	10.0	67.6	1.4

Silicon and Sulphur in Basic Pig Iron

It has been known generally that silicon and sulphur are affected by furnace temperatures. This is in agreement with laboratory tests on the relative desulphurizing power of blast-furnace slags. The oxide investigation, however, afforded an opportunity to obtain quantitative data under actual operating conditions. The variation in silicon and sulphur with temperature fluctuations from cast to cast was obtained and corrections were made for small changes in basicity. A definite relation between silicon, sulphur, and the temperature of the iron was established, as shown graphically in figure 21. In general, a variation of 40° F. in the average temperature of the slag in the runner resulted in a 10-point (0.10-percent) variation in silicon and a 1-point (0.01-percent) variation in sulphur. Temperature was measured mainly with a noble-metal thermocouple placed to a depth of 8 to 10 inches in the metal in the runner.

METALLURGY OF STEEL SECTION

Magnetic Properties of Steel

The work on the magnetic properties of slags has been continued during the past year. Slags from other steel plants have been examined and found to contain a much smaller quantity of iron particles and also to be inherently much less magnetic than the series described in the previous annual report. To carry out the magnetic separation it was necessary to make a simple alteration of the apparatus to handle material of lower magnetic susceptibility. The tendency of the powdered slags to separate into portions of varying susceptibility was less pronounced than in the previous case. As a result of these experiments, it was concluded that the susceptibility of the slag is more sensitive to composition than had been realized. Possibly the kind of furnace atmosphere has some effect.

The major activity, however, in connection with the magnetic properties of slags was the development of equipment for determining susceptibility at elevated temperatures. Slags contain a variety of ferrites that are weakly ferromagnetic, and since their Curie temperatures (temperature above which the substance is no longer ferromagnetic) are quite different, it appeared possible that a knowledge of the points at which there was a decrease in susceptibility would aid in identifying the constituents.

To conduct such experiments it was necessary to have an electromagnet with the proper type of magnetic field and of sufficient size to permit placing a small electric furnace between the pole faces. Such a magnet was built with the aid of a pair of pole pieces designed and constructed by the S. G. Frantz Co. Figure 22 is a photograph of this magnet rotated through 90° from its usual position to show the nature of the gap. The dimensions of the gap are such that a small, water-jacketed, cylindrical furnace, 1-1/8-inch outside diameter, can be placed between the pole faces. With this furnace the sample can be heated readily to 800° C., and a measurement made at any time of the amount of pull on the sample produced by the nonuniform field of the magnet.

A series of slags from an 0.08-percent carbon steel from the Republic Steel Corporation was investigated with this apparatus. The first experiments were made by placing the powdered samples in a silica capsule, exhausting the air, and sealing off the connecting tube. The results of these measurements are shown in figure 23, which includes specimens of slag taken from the furnace during the last 4-1/2 hours of the manufacturing process. The steps show distinct decreases in susceptibility of the sample, which are believed to be caused by a ferromagnetic constituent having passed through its Curie point. In general, the position of the steps with respect to temperature is so variable as to suggest a continuous change in the nature of the constituents with a corresponding shift of Curie temperature.

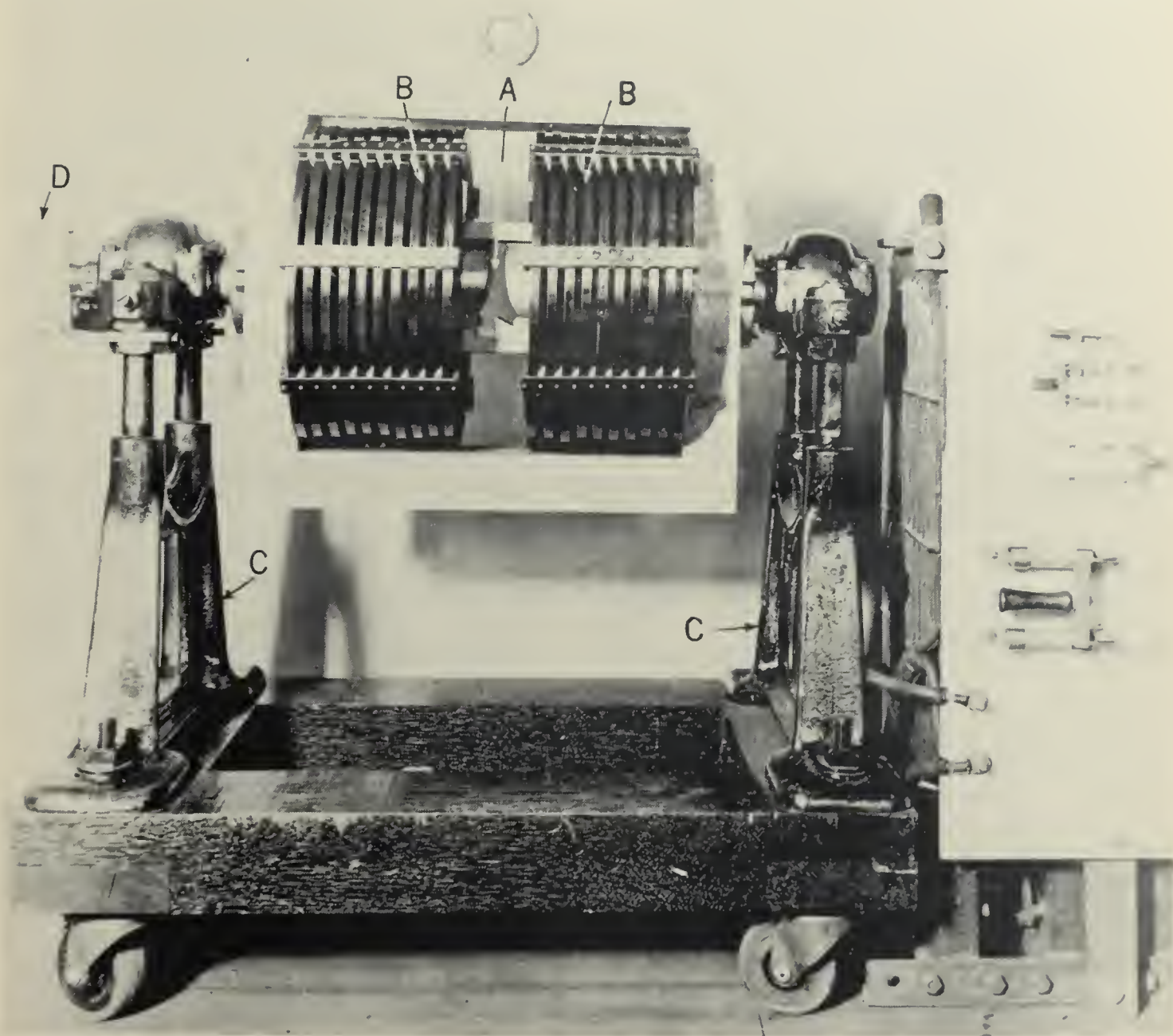


Figure 22.—The electromagnet for experiments on the Curie point of slags.

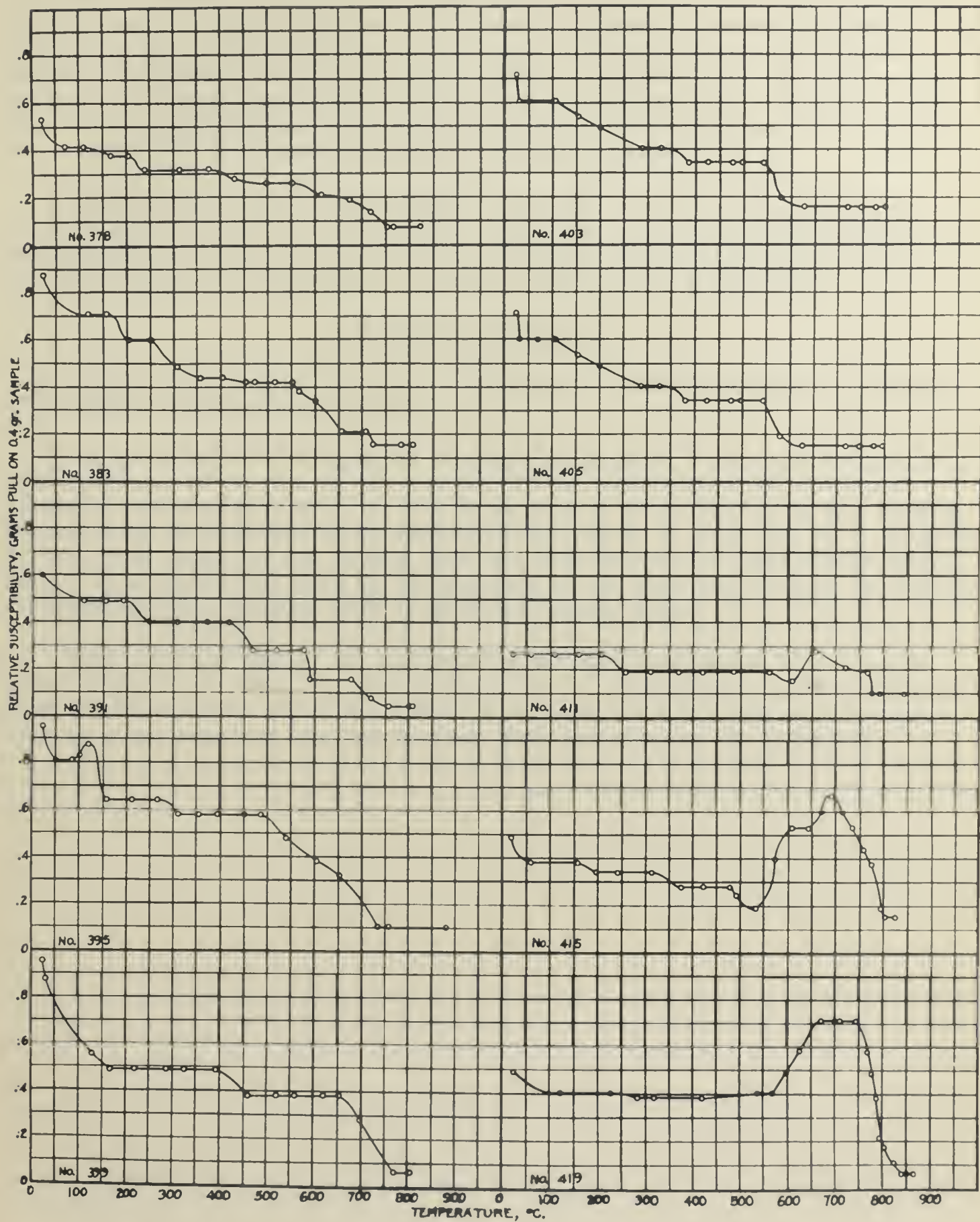


Figure 23.—Relative susceptibility-temperature curves for a series of basic open-hearth slags.

The presence of magnetite in samples 391, 403, and 405 is indicated by the drop in susceptibility just below 600° C. Iron is obviously present in all the samples except 403 and 405, since a decrease in susceptibility is to be observed well above 600° C. The tendency of this step to occur at higher temperatures as the heat progressed is believed to be related to the increase of Curie temperature of iron with decreasing carbon content. The rise of susceptibility at high temperatures of the last three slags appears to be caused by the decomposition of some constituent.

Experiments with this same series of slags, but with the samples exposed to an atmosphere of oxygen failed to show as much differentiation between samples as the tests in vacuo. Similar trials in hydrogen atmosphere have not advanced sufficiently to permit drawing any conclusions.

Metal-Slag Emulsions

Various experiments have been performed in connection with the problem of producing metal-slag emulsions.

The coercimeter developed in the Special-Studies Section is being investigated as to its use in the control of the manufacture and treatment of steel. An article entitled "The Mechanism of Steel Hardening and Drawing as indicated by Coercive Force Measurements", by R. S. Dean and C. Y. Clayton, is the first paper in the series that has been written as data become available. This report was read at the meeting of the American Society for Metals in Atlantic City, October 1937 -- National Metals Week.

Work is under way on the application of this instrument to the rapid determinations of carbon in open-hearth steel. Preliminary results indicate that the coercimeter will be a useful tool in steel mills that can be operated by a nontechnical man who has had some training in its operation.

AFTER THIS REPORT HAS SERVED YOUR PURPOSE AND IF YOU HAVE NO FURTHER NEED FOR IT, PLEASE RETURN IT TO THE BUREAU OF MINES. THE USE OF THIS MAILING LABEL TO DO SO WILL BE OFFICIAL BUSINESS AND NO POSTAGE STAMPS WILL BE REQUIRED.

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UNITED STATES
DEPARTMENT OF THE INTERIOR
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BUREAU OF MINES
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REPORT OF INVESTIGATIONS

ANALYSES OF CRUDE OILS FROM SOME OF THE MORE RECENTLY DISCOVERED ROCKY MOUNTAIN FIELDS

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WALTER MURPHY AND H. M. THORNE

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Report of Investigations 3358. Analyses of Crude Oils from Some of the More Recently Discovered Rocky Mountain Fields, by Walter Murphy and H. M. Thorne.

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REPORT OF INVESTIGATIONS

UNITED STATES DEPARTMENT OF THE INTERIOR - BUREAU OF MINES

ANALYSES OF CRUDE OILS FROM SOME OF THE MORE RECENTLY DISCOVERED ROCKY
MOUNTAIN FIELDS^{1/}

By Walter Murphy^{2/} and H. M. Thorne^{3/}

INTRODUCTION

For a number of years the Bureau of Mines has been studying the properties of crude petroleums produced in the United States and the Western Hemisphere. A number of reports have been published giving the analyses of crudes from the different fields and regions as made by the Bureau of Mines Hempel method of analysis.^{4/} Several of these reports include analyses of crude oils from the Rocky Mountain region.^{5/}

The search for new fields and new producing horizons in old fields in the Rocky Mountain region has been quite intensive during the last few years, and a number of discoveries have been made.

The Lance Creek field in eastern Wyoming, which for a number of years produced from the Dakota and Muddy sands, recently was drilled to the Sundance sands, and production from these now is much greater than from the old producing horizons. In the Rock Creek field a number of wells have been deepened to the Sundance sand, increasing the production of this field. The Medicine Bow field, which was discovered in 1935, also produces from the Sundance sand. Two discoveries have been made in Moffat County, Colo. - the Hiaawatha and Powder Wash fields. Production from these fields is from the Wasatch formation of the Tertiary.

^{1/} The Bureau of Mines will welcome reprinting of this paper provided the following footnote acknowledgment is used: "Reprinted from Bureau of Mines Report of Investigations 3358."

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^{4/} Smith, N. A. C., and Lane, E. C., Tabulated Analyses of Representative Crude Petroleums of the United States: Bull. 291, Bureau of Mines, 1928, 69 pp.

^{5/} Thorne, H. M., and Murphy, Walter, A Survey of the High-Sulphur Crude Oils (Black Oils) Produced in Wyoming: Tech. Paper 358, Bureau of Mines, 1932, 56 pp.

Dean, E. W., Cooke, M. B., and Bauer, A. D., Properties of Typical Crude Oils from the Producing Fields of the Rocky Mountain District: Rept. of Investigations 2235, Bureau of Mines, 1921, 50 pp.

Rue, H. P., Properties of Crude Oil from the Greasewood Flat Area in Colorado: Rept. of Investigations 3174, Bureau of Mines, 1932, 3 pp.

Analyses of crude oils from several of the more recently discovered Rocky Mountain fields and from new producing horizons in old fields are shown in this paper.

ACKNOWLEDGMENTS

This report was made under the general supervision of R. A. Cattell, chief engineer, Petroleum and Natural Gas Division, Bureau of Mines, Washington, D. C., and under the direct supervision of H. P. Rue, supervising engineer, Petroleum Experiment Station, Bureau of Mines, Laramie, Wyo.

The authors are grateful to the oil-producing companies for their cooperation and assistance in the collection of samples and in supplying production data. Samples were analyzed by the personnel of the Bureau of Mines Petroleum Experiment Station, Laramie, Wyo.

HISTORY AND PRODUCTION DATA

The samples were collected and production data were obtained by Bureau engineers. The histories of the fields were obtained from various authorities.

Cut Bank

The Cut Bank oil and gas field is in Glacier County, northwestern Montana, about 20 miles from the Canadian border and about 12 miles west of the Kevin-Sunburst gas field. This field is the source of supply for the Montana Power Co. gas line to Butte, Anaconda, Deer Lodge, and Helena. The discovery well for gas, No. 1 Michaels in the center of the $SE\frac{1}{4}$ $SE\frac{1}{4}$ sec. 28, T. 36 N., R. 5 W., was completed in 1930 and produced 15,000,000 cubic feet of gas at a depth of 2,750 to 2,771 feet.^{6/}

The discovery well for oil, the Brecken and Hatch Oil Co. No. 1 Haines in the $W\frac{1}{2}$ $SE\frac{1}{4}$ $SE\frac{1}{4}$ sec. 2, T. 34 N., R. 6 W., was completed in June 1931 at a depth of 2,760 to 2,762 feet in what is probably the Sunburst sand. This well had an initial production of 50 barrels per day.^{7,8/}

According to Dobbin and Erdmann,^{9/} "The basal sand of the district is the Cut Bank sand, which seems to be the direct equivalent of the Cosmos sand of the Border district and is the principal producer."

Present production from the field is about 9,000 barrels daily. Total production from date of discovery to December 31, 1936, was 7,015,433 barrels.

^{6/} Oil and Gas Jour., vol. 30, no. 2, May 28, 1931, p. 82.

^{7/} Oil and Gas Jour., vol. 29, no. 40, Feb. 19, 1931, p. 43.

^{8/} Oil and Gas Jour., vol. 30, no. 1, May 21, 1931, p. 133.

^{9/} Am. Assoc. Petrol. Geol., Problems of Petroleum Geology: Tulsa, Okla., 1934, pp. 711-712.

Hiawatha

The Hiawatha field is in T. 12 N., Rs. 100 W and 101 W., Moffat County, Colo., and in T. 12 N., R. 100 W., Sweetwater County, Wyo. The discovery well, F. L. Wilson No. 1, was drilled by Mountain Fuel Supply Co. in the center of sec. 22, T. 12 N., R. 100 W., in Colorado and was completed as a gas well in October 1926 at a depth of 2,215 to 2,238 feet in a sand in the Wasatch series. It was not put on production until October 1929, at which time a potential production of 45 million cubic feet of gas at 840 pounds per square inch rock pressure was obtained. This well soon started producing 8 to 15 barrels of oil per day with each million cubic feet of gas. According to Nightingale,^{10/} the decline in pressure due to the production of gas caused an encroachment of crude oil into the well.

Gas from this field and from Baxter Basin is marketed in Rock Springs, Wyo. and in Salt Lake City and Ogden, Utah. To date, no oil from this field has been marketed.

Lance Creek

The Lance Creek field is in Tps. 35 N. and 36 N., Rs. 64 W. and 65 W., Niobrara County, eastern Wyoming, about 30 miles northwest of the town of Lusk. According to Nowels and Dewees,^{11/} the discovery well in this field was drilled by the Ohio Oil Co. in the extreme northwest corner of sec. 36, T. 36 N., R. 65 W. In March 1918 this well produced 80 barrels of oil per 24 hours from the Newcastle sand (Muddy) at 2,689 feet. It was deepened later and was completed in October 1918 for 1,500 barrels of oil per day from the second Dakota sand at 3,663 feet. Geological work and additional drilling during 1919 and 1920 showed that the apex of the structure was situated about $3\frac{3}{4}$ miles west and 1 mile south of the discovery well in the south half of sec. 32, T. 36 N., R. 65 W.

When the discovery well came in it was thought that a field of great importance had been found. However, this well began to produce considerable water about July of the following year and shortly after was abandoned. Subsequent drilling of other wells met with varying success; a well producing clean oil, free from water, was the exception rather than the rule.

The first Sundance sand was proved for production by the Ohio Oil Co. in the fall of 1930. In March 1935 the same company found oil in the basal Sundance in well No. 2, Agnes Rohlff, 100 feet south of the center of the NE $\frac{1}{4}$ SE $\frac{1}{4}$ sec. 32, T. 36 N., R. 65 W. This well had an initial production of over 2,000 barrels of oil of 47.5° A.P.I. gravity

^{10/} Am. Assoc. Petrol. Geol., Geology of Natural Gas: Tulsa, Okla., 1935, pp. 341-361.

^{11/} Nowels, K. B. and Dewees, E. J., Petroleum Engineering in the Lance Creek Field, Niobrara County, Wyo.: Bureau of Mines in Cooperation with the University of Wyo., January 1926.

with 3,000,000 cubic feet of gas from the third bench of the Sundance at a depth of 3,843 to 3,901 feet.^{12/} Recent development indicates that the productive area of the Sundance sand will be greater than that of the upper formations.

On December 31, 1936, 73 wells were capable of producing in the Lance Creek field. Of these, 13 oil and 12 gas wells were in the Muddy and Dakota sands. Most of these and the two oil wells in the first Sundance sand were shut in. Forty-six oil wells were producing from the basal Sundance sand. All of the present production of 12,000 barrels per day, with the exception of about 200 barrels from the Muddy and Dakota sands, comes from the basal Sundance sand.

Total production from this field from date of discovery to December 31, 1936, was 6,984,630 barrels.

Medicine Bow

The Medicine Bow field is in T. 21 N., R. 79 W., east-central Carbon County, Wyo. The discovery well in the field, William Kyle No. 1, in the NE $\frac{1}{4}$ SE $\frac{1}{4}$ NE $\frac{1}{4}$ of sec. 26, T. 21 N., R. 79 W., was completed in June 1935 by the Ohio Oil Co. and the California Co. The first bench of the Sundance sand contained gas and the second bench oil and gas. The well was completed so that the two horizons may be produced independently of each other. The initial production tests established a daily potential of 81 million cubic feet of gas from the first bench, at 5,157 to 5,207 feet, and 6,360 barrels of oil from the second bench of the Sundance sand, at 5,299 to 5,400 feet.

The field now has five producing wells and has produced 163,129 barrels of oil up to December 31, 1936. The average daily production for the field at present (May 1937) is 3,000 barrels.

Pondera

The Pondera field is in Pondera County, northwestern Montana, about 40 miles south of the Kevin-Sunburst field. The discovery well, located in the center of the SE $\frac{1}{4}$ SE $\frac{1}{4}$ sec. 17, T. 27 N., R. 4 W., was completed by the Montana Pacific Oil Co. in June 1927. This well did not develop oil in commercial quantities, although a showing of oil was obtained in the upper part of the Madison limestone at a depth of 2,072 feet.

In March 1928 the Fulton Petroleum Co. completed its No. 1 State in the SW $\frac{1}{4}$ SE $\frac{1}{4}$ NW $\frac{1}{4}$ sec. 16, T. 27 N., R. 4 W., about $\frac{1}{2}$ mile north-east of the discovery well. This well had an initial daily production of 1,000,000 cubic feet of gas and 100 barrels of oil from the upper part

^{12/} Brainerd, A. E., and Lavington, C. S., The Lance Creek Oil and Gas Field, Niobrara County, Wyo.: Mines Mag., Colorado School of Mines Alumni Assoc., vol. 26, no. 2, February 1936, p. 15.

of the Madison limestone at a depth of 2,042 feet.^{13/}

The present daily production is about 1,100 barrels. The total production during 1936 was 442,400 barrels.

Powder Wash

The Powder Wash field is in Moffat County, northwestern Colorado. This structure was proved for gas production in April 1931, when well No. 1 Musser, in the center of the NE $\frac{1}{4}$ sec. 5, T. 11 N., R. 97 W., developed 34,000,000 cubic feet of gas in the Upper Wasatch formation at a depth of 2,152 feet.^{14/}

The Mountain Fuel Supply Co. completed its Allen No. 1, in the NE $\frac{1}{4}$ SW $\frac{1}{4}$ SE $\frac{1}{4}$ sec. 32, T. 12 N., R. 97 W., in November 1936 in the lower Wasatch formation at a depth of 5,012 to 5,032 feet. This well produced 1,120 barrels of oil and 5,000,000 cubic feet of gas through the 2 $\frac{1}{2}$ -inch tubing on a 24-hour production test.

Quealy Dome

The Quealy Dome field is in sec. 13, T. 17 N., R. 77 W., and secs. 18 and 19, T. 17 N., R. 76 W., southwestern Albany County, Wyo., about 25 miles northwest of the city of Laramie. The discovery well, No. 1 Holst in the NE $\frac{1}{4}$ SE $\frac{1}{4}$ SE $\frac{1}{4}$ sec. 13, T. 17 N., R. 77 W., was drilled in 1934 by the California Co. and obtained a production of 50 barrels per day from the Muddy sand at a depth of 3,146 to 3,165 feet. This well was deepened, and in April 1935 oil was discovered in the Dakota sand at 3,231 to 3,263 feet. On a 10-day test, this well pumped 600 barrels per day from the Dakota sand.^{15/}

One well was drilled to the Sundance sand, where a production of 150 barrels of oil per day was obtained. This well was plugged back and completed in the Dakota sand.

In the latter part of June 1937 there were eight wells in this field with an initial production of 1,300 barrels per day from the Dakota sand. The total production for 1936 was 80,147 barrels.

Rock Creek

The Rock Creek field, also known as the Rock River field, is in Tps. 19 N. and 20 N., R. 78 W., in the southeastern part of Carbon County, Wyo. Oil was discovered on this structure in May 1918, when the Ohio Oil Co. obtained an initial production of 50 barrels per day in the

^{13/} Bull. Am. Assoc. Petrol. Geol., vol. 13, no. 7, July 1929, pp. 782-783.

^{14/} Inland Oil Index, vol. 34, no. 22, November 27, 1936.

^{15/} Schoenfelt, C. E., Résumé, Rocky Mountain Oil and Gas Operations for 1935: Petroleum Information Inc., Denver, Colo.

Muddy sand at 2,581 to 2,609 feet. This field continued to produce from the Dakota and Lakota sands (known locally as Muddy and Dakota, respectively), and in June 1935 an initial production of 200 barrels per day was obtained from the Sundance sand at 3,096 to 3,200 feet by deepening one of the wells. In May 1937 there were 42 producing wells in the field, 35 of which produce from the first and second Muddy and Dakota sands and seven from the Sundance sand. The daily production for the field at present is about 1,200 barrels.

The total production from the field from date of discovery to December 31, 1936, was 16,706,934 barrels.

Waugh Dome

The Waugh Dome field is in sec. 7, T. 44 N., R. 96 W., and secs. 1, 2, and 12, T. 44 N., R. 97 W., Hot Springs County, northeastern Wyoming, about 15 miles northeast of Thermopolis. The discovery well, the California Exploration Co. No. 1 Government in the NE $\frac{1}{4}$ sec. 12, T. 44 N., R. 97 W., was completed in December 1934. This well had an initial production of 583 barrels per day on the pump from the Embar limestone at a depth of 3,780 to 3,807 feet. The present daily production is about 300 barrels. The total production from this field from date of discovery to December 31, 1936, was 99,795 barrels.

Wertz Dome

The Wertz field is in T. 26 N., Rs. 89 W. and 90 W., Carbon and Sweetwater Counties, Wyo., adjoining the Little Lost Soldier field. Gas was discovered in this structure in the Dakota sand in 1920. An initial production 42,868,000 cubic feet of gas was obtained from the Dakota sand at 3,427 to 3,435 feet with a rock pressure of 1,800 pounds per square inch. Later, two gas wells were completed in the Frontier formation at 2,160 to 2,265 feet for 6 to 7 million cubic feet per day. At present there are seven gas wells in the field producing from the Dakota, Lakota, and Sundance sands.

Oil was discovered in the Tensleep sand at 5,872 to 5,886 feet in December 1936 by the Sinclair-Wyoming Oil Co. The discovery well, No. 10-A Wertz, in the SE $\frac{1}{4}$ NW $\frac{1}{4}$ sec. 7, T. 26 N., R. 89 W., had an initial production of 1,350 barrels of oil per day.^{16/}

This field has produced 59,524,057,000 cubic feet of gas from date of discovery to December 31, 1936.

CHARACTERISTICS OF THE CRUDE OILS

The Hempel analyses shown in this paper were made at the Bureau of Mines Petroleum Experiment Station laboratory in Laramie, Wyo. The altitude of Laramie is 7,200 feet, with an average barometric pressure

of 530 mm of mercury. This low barometric pressure influences the yield of distillate obtained on air distillation. More distillate will be obtained to a specified temperature at 580 mm pressure than at higher barometric pressures. The boiling points shown in these analyses have not been converted to 760 mm, for it is believed that the yields as shown will more nearly approach refinery yields in the Rocky Mountain region.

Several formulas and charts^{17/} have been developed for determining the boiling points of individual hydrocarbons or fractions at any desired pressure when their boiling points at one pressure is known. The American Society for Testing Materials, in its Standard Method of Test for Distillation of Gasoline, Naphtha, Kerosene, and Similar Petroleum Products (D86-35), states: "The actual barometric pressure shall be ascertained and recorded, but no correction shall be made except in case of disputes. In such cases the temperature points shall be corrected to 760 mm (29.92 in.) by the use of the Sydney Young equation, as follows:

For centigrade readings:

$$C_c = 0.00012 (760-P) (273 + t_c)$$

For Fahrenheit readings:

$$C_f = 0.00012 (760-P) (460 + t_f)$$

in which C_c and C_f are, respectively, corrections to be added to the observed temperature t_c or t_f , and P is the actual barometric pressure in millimeters of mercury."^{13/}

A comparison of the boiling-point conversion data of the different investigators indicates that the Sydney Young equation gives values that are sufficiently accurate for most practical purposes when

^{17/} Wilson, O. G., Jr., Vapor-Pressure Chart for Paraffin Hydrocarbons: Ind. and Eng. Chem., vol. 20, no. 12, December 1928, p. 1363.

Coats, H. B., and Brown, G. G., A Vapor-Pressure Chart for Hydrocarbons: Univ. of Michigan, Dept. of Eng. Research Circ. 2, December 1928, 17 pp., 1 chart.

Copson, R. L., and Frolich, Per K., Vapor-Pressure Chart for Lower Aliphatic Hydrocarbons: Ind. and Eng. Chem., vol. 21, no. 11, November 1929, p. 1116.

Watson, K. M., and Wirth, Chas., III, Routine High-Vacuum Distillation of Oils; An Apparatus and Conversion Chart: Ind. and Eng. Chem., Anal. Ed., vol. 7 no. 1, Jan. 15, 1935, p. 72.

Beale, E. S. L., and Docksey, P., A Wide-Range Boiling-Point-Conversion Chart for Hydrocarbons and Petroleum Products: Jour. Inst. Petrol. Technol., vol. 21, 1935, pp. 860-870.

^{18/} American Society for Testing Materials, A. S. T. M., Standards on Petroleum Products and Lubricants: Prepared by Committee D-2, Philadelphia, Pa., September 1936, pp. 96-104.

the barometric pressure is between 760 and 500 mm. However, at pressures below 500 mm the divergence of the data obtained by Young's equation from that obtained from the equations of the other investigators becomes so great that the equation seems of little value.

Cut Bank Crude

The crude oil from the Cut Bank field, Montana, is of intermediate base and is wax-bearing, as indicated by the Bureau of Mines interpretation of analyses.^{19/} It contains 32.7 percent gasoline and naphtha, 5.3 percent kerosene distillate, 21.0 percent gas oil, and 17.5 percent lubricating distillates, as shown by Hempel analysis. The sulphur and carbon residue contents of the crude oil are fairly high, indicating the presence of asphalt. A rough estimate of the asphalt content of a crude oil may be made by multiplying its percent carbon residue by 2.5.^{20/} Cut Bank crude contains 6.25 percent asphalt, as indicated by this method. (See sample 36L-11, p. 13.)

Hiawatha Crude

The crude oil produced from the Hiawatha field is of intermediate-paraffin base; that is, the specific gravity-boiling point relationship for the light fractions is similar to that of intermediate base crudes, and the specific gravity-boiling point relationship for the heavy fractions is similar to that of paraffin-base crudes. The crude has a high wax content, as indicated by its high pour point (75°F.) and the high cloud points of the vacuum fractions. This high pour point makes transportation of the crude difficult, particularly in cold weather. The Bureau of Mines Hempel analysis shows the crude oil to contain 19.8 percent of gasoline and naphtha, no kerosene distillate (the distillate boiling within the kerosene range has a specific gravity greater than 0.825), 35.4 percent gas oil, and 23.2 percent lubricating distillates. The carbon residue and the sulphur contents are low. (See sample 36L-3, p. 14.)

Lance Creek Crude Oils

This report includes analyses of oils from the three principal producing horizons in the Lance Creek field. The oil production from the field was principally from the Dakota sand. The oil from this formation is of intermediate-paraffin base, is wax-bearing, and contains 29.8 percent gasoline and naphtha, 5.3 percent kerosene distillate, 21.2 percent gas oil, and 25.5 percent lubricating distillates. The sulphur and carbon residue contents are low. (See sample 36L-9, p. 15.)

The oil produced from the upper Sundance sand is of paraffin

^{19/} Lane, E. C., and Garton, E. L., Base of a Crude Oil: Rept. of Investigations 3279, Bureau of Mines, September 1935, 12 pp.

^{20/} See footnote 4, p. 1, Bull. 291, p. 8.

base and is wax-bearing, but it contains considerable asphalt for this type of oil, as indicated by its carbon residue and color. The crude contains 36.2 percent gasoline and naphtha, 18.5 percent kerosene distillate, 7.4 percent gas oil, and 16.4 percent lubricating distillates. Its sulphur content is low. (See sample 36L-7, p. 16.)

The oil produced from the basal Sundance sand is classed as wax-bearing and of paraffin-intermediate base. It contains 46.2 percent gasoline and naphtha, 17.6 percent kerosene distillate, 7.3 percent gas oil, and 15.3 percent lubricating distillates. It has a low sulphur and carbon residue content. The crudes from the Sundance sands have similar characteristics, except that the oil from the basal part of the sand contains 10 percent more gasoline and naphtha and only about one-fifth as much asphalt as that from the upper. The basal Sundance is the principal producing horizon in the Lance Creek field at the present time. (See sample 36L-8, p. 17.)

Medicine Bow Crude

The Medicine Bow field produces an extremely light gravity, low sulphur content crude oil of paraffin-intermediate base and wax-bearing. It contains 71.2 percent gasoline and naphtha, 12.5 percent kerosene distillate, 1.8 percent gas oil, and 3.9 percent lubricating distillates. The crude, being deficient in heavy ends, would have little value in the manufacture of lubricating oil, but the small amount of lubricating distillate obtained should be of good quality. (See sample 37L-11, p. 18.)

Pondera Crude

The Pondera field, Montana, produces an intermediate-base wax-bearing crude. It contains 30.9 percent gasoline and naphtha, 4.8 percent kerosene distillate, 17.7 percent gas oil, and 20.1 percent lubricating distillates. It has a high asphalt content, as indicated by its high carbon residue and high sulphur contents and by its color. It is similar to the crude from the Cut Bank field about 50 miles to the northwest, except that the Pondera sample contains more sulphur and asphalt and slightly less gasoline. (See sample 36L-10, p. 19.)

Powder Wash Crude

The crude oil from the Powder Wash field, Colorado, is of intermediate-paraffin base and is wax-bearing. In fact, it contains so much wax that it is difficult to handle in cold weather. Its pour point is 85°F. The crude contains 22.1 percent gasoline and naphtha, 6.1 percent kerosene distillate, 26.5 percent gas oil, and 26.6 percent lubricating distillates. It has a low sulphur and carbon residue content, indicating lubricating oil possibilities. The Powder Wash and Hiawatha fields are in the same locality, and both crudes are produced from the Wasatch of the Tertiary. The oils from the two fields are similar, yielding approximately the same amount of products having the same characteristics, except that the Hiawatha crude contains about twice as much sulphur and asphalt

as that from Powder Wash. However, both the sulphur contents and the asphalt contents are low and of no particular consequence. (See sample 36L-18, p. 20.)

Quealy Dome Crude

The crude oil produced from the Dakota sand in the Quealy Dome field is of intermediate base and wax-bearing. It contains 24.8 percent gasoline and naphtha, 4.4 percent kerosene distillate, 18.3 percent gas oil, and 24.2 percent lubricating distillates. The sulphur content is low, but the carbon-residue value indicates an asphalt content of 6.5 percent. (See sample 36L-14, p. 21.)

Rock Creek Crude

The crude oil produced from the Sundance sand in the Rock Creek field is of intermediate base and wax-bearing. It contains 31.4 percent gasoline and naphtha, 4.4 percent kerosene distillate, 18 percent gas oil, and 20.4 percent lubricating distillates. Its sulphur content is low, but it contains some asphalt, as indicated by its carbon residue. The oil from the Sundance sand is almost identical with that from the upper producing sands (Muddy and Dakota) in this field.^{21/} (See sample 36L-16, p. 22.)

Waugh Dome Crude

The crude oil produced from the Embar limestone in the Waugh Dome field is a "black",^{22/} as indicated by its high sulphur and asphalt content, and is classified as intermediate-base and wax-bearing. It contains 19.9 percent of gasoline and naphtha, 4.6 percent kerosene distillate, 19.8 percent gas oil, and 24.2 percent lubricating distillates. The crude oil contains 1.77 percent sulphur and has a carbon-residue content of 6.4 percent. These properties indicate that the oil would be more suitable for manufacturing road oils than lubricants. (See sample 36L-6, p. 23.)

Wertz Crude

The oil produced from the Wertz field is classified as intermediate-base, wax-bearing, and because of its high sulphur and asphalt content is called a "black oil". It contains 30.8 percent gasoline and naphtha, 5.1 percent kerosene distillate, 18.5 percent gas oil, and 22.0 percent lubricating distillate. The oil, although known as a "black oil", is of the higher type of this class, as it contains considerable gasoline and has a sulphur content of 1.3 percent, as compared with 2 to 4 percent for most of the Wyoming "black oils." Its carbon residue is also lower than that of most Wyoming "black oils." The crude oil from this field is similar to those from the Tensleep sands in the adjoining Little

^{21/} See footnote 4: Sample 314, Bull. 291.

^{22/} See footnote 5: Tech. Paper 538.

Lost Soldier field and the nearby Mahoney Dome field.^{23/} (See sample 37L-5, p. 24.)

CRUDES FROM SUNDANCE SANDS

Four of the crude oils included in this report are produced from the Sundance sands. In Lance Creek and Rock Creek, both old fields with production in the Muddy and Dakota sands, oil has been discovered recently in the Sundance sands. The Medicine Bow field, a recent discovery, produces from the Sundance sands.

On first inspection, the oils produced from the Sundance sands in the different fields appear quite different because of the wide variation in their gravity and gasoline content. However, a comparison of their other characteristics reveals that they are basically similar. The sulphur contents of the two Lance Creek samples and the Medicine Bow sample are less than 0.1 percent, while that of the Rock Creek sample is only 0.16 percent. The carbon residues of the four crude samples range from less than 0.1 percent to 2.2 percent. This difference in carbon residue probably is due to different amounts of asphaltic material dissolved in the crude. A difference of about 5.5 percent of dissolved asphalt would account for the carbon residue variation.

The A. P. I. gravity of "key fractions no. 1" (fractions distilling between 482°F. and 527°F. at atmospheric pressure) are 40.4, 40.2, 40.0, and 37.0, and of "key fractions no. 2" (fractions distilling between 527°F. and 572°F. at 40 mm pressure) are 30.0, 29.8, 29.8, and 26.6 for the Lance Creek upper Sundance, Lance Creek basal Sundance, Medicine Bow, and Rock Creek samples, respectively. The gravities of these "key fractions" are used by the Bureau of Mines in determining the "base" of crude oils.^{24/}

A. P. I. gravities for "key fraction no. 1" of 40.0 or higher show the lower-boiling fractions of the oil to be paraffinic in character. If the gravity is lower than 40.0 but higher than 33.0, the lower-boiling fractions are intermediate, but if 33.0 or lower they are naphthenic. The characteristics of the higher-boiling fractions are determined by the gravity of "key fraction no. 2." If the A. P. I. gravity is 30.0 or higher they are paraffinic; below 30.0, but higher than 20.0, intermediate; and 20.0 or below, naphthenic. All of the crude oils included in this report that are produced from the Sundance sands are either paraffinic or intermediate in character, with properties falling near the dividing line between the two classes of crudes.

Table 1 compares some of the properties of the four crude oils produced from the Sundance sands.

^{23/} See footnote 5: Tech. Paper 538, pp. 25 and 26.

^{24/} See footnote 19, p. 15.

TABLE 1. - Summary of properties of crude oils produced from Sundance sands in
Wyoming

Field-----	Lance Creek	Lance Creek	Medicine Bow	Rock Creek
County-----	Niobrara	Niobrara	Carbon	Carbon
Producing formation-----	Upper Sundance	Basal Sundance	Sundance	Sundance
Depth, feet-----	3,624	3,880	5,520	3,188
Properties of Crude:				
Gravity, A.P.I.-----	42.8	49.4	63.4	36.0
Sulphur, percent-----	Less than 0.1	Less than 0.1	Less than 0.1	0.16
Carbon residue, percent	2.2	0.4	Less than 0.1	2.1
Viscosity, seconds-----	46 at 70°F.	34 at 70°F.	30 at 70°F.	43 at 100°F.
Pour point, °F.-----	10	Below 5	Below 5	20
Gasoline and naphtha:				
Percent-----	36.2	46.2	71.2	31.4
Gravity, A. P. I.-----	63.1	63.9	66.7	58.9
Vacuum fractions:				
437°-482°F., percent---	5.6	4.6	1.4	4.8
Viscosity at 100°F.---	56	57	57	60
482°-527°F., percent---	4.7	3.7	0.3	5.5
Viscosity at 100°F.---	72	78	64	83
527°-572°F., percent---	5.8	6.2	1.6	7.7
Viscosity at 100°F.---	129	142	101	171
Residuum:				
Percent-----	17.9	8.8	1.0	23.8
Gravity, A. P. I.-----	17.1	20.9	19.4	15.9
Carbon residue, percent	10.6	4.2	4.1	7.8

REPORT OF CRUDE PETROLEUM ANALYSIS

Sample 36L-11

Lukens No. 2
2,935 feet
R. C. Jefferies

Cut Bank field
Cut Bank sand
Samples, May 1936

Montana
Glacier County
Sec. 12, T. 33 N., R. 6 W.

GENERAL CHARACTERISTICS

Specific gravity, 0.840

Percent sulphur, 1.01

Saybolt Universal viscosity at 100°F., 41 sec.

A.P.I. gravity, 37.0°

Pour point, below 5°F.

Color, greenish-black

DISTILLATION, BUREAU OF MINES, HEMPEL METHOD

Dry distillation Barometer, 591 mm First drop: 29°C. (84°F.)

Temperature, °C.	Per- cent cut	Sum, per- cent	Sp.gr. of cut	°A.P.I. of cut	Viscosity at 100°F.	Cloud test, °F.	Temperature, °F.
Up to 50	4.3	4.3	0.670	79.7			Up to 122
50-75	2.1	6.4	.677	77.5			122 - 167
75-100	4.2	10.6	.713	66.9			167 - 212
100-125	5.2	15.8	.741	59.4			212 - 257
125-150	5.9	21.7	.768	52.7			257 - 302
150-175	5.5	27.2	.783	49.2			302 - 347
175-200	5.5	32.7	.798	45.8			347 - 392
200-225	5.3	38.0	.813	42.5			392 - 437
225-250	6.0	44.0	.829	39.2			437 - 482
250-275	6.4	50.4	.845	36.0			482 - 527

Vacuum distillation at 40 mm

Up to 200	6.8	6.8	0.855	34.0	44	25	Up to 392
200 - 225	2.2	9.0	.872	30.8	48	30	392 - 437
225 - 250	5.9	14.9	.885	28.4	61	50	437 - 482
250 - 275	5.7	20.6	.899	25.9	94	70	482 - 527
275 - 300	5.5	26.1	.909	24.2	173	90	527 - 572

Carbon residue of residuum, 10.4 percent. Carbon residue of crude, 2.5 percent.

APPROXIMATE SUMMARY

	Percent	Sp.gr.	°A.P.I.	Viscosity
Light gasoline	10.6	0.688	74.2	
Total gasoline and naphtha	32.7	.746	58.2	
Kerosene distillate	5.3	.813	42.5	
Gas oil	21.0	.846	35.8	
Nonviscous lubricating distillate	9.7	.874-.901	30.4-25.5	50 - 100
Medium lubricating distillate	7.1	.901-.926	25.5-21.3	100 - 200
Viscous lubricating distillate	.7	.926-.928	21.3-21.0	Above 200
Residuum	20.8	.973	13.9	
Distillation loss	2.7			

REPORT OF CRUDE PETROLEUM ANALYSIS

Sample 36L-3

Wilson No. 1	Hiawatha field	Colorado
2,280 feet	Wasatch formation	Moffat County
Mountain Fuel Supply Co.	Tertiary	Sec. 22, T. 12 N., R. 100 W
	Sampled, March 1936	

GENERAL CHARACTERISTICS

Specific gravity, 0.850	A.P.I. gravity, 35.00
Percent sulphur, 0.21	Pour point, 75°F.
Saybolt Universal viscosity at 100°F., 43 sec.	Color, medium green

DISTILLATION, BUREAU OF MINES, HEMPEL METHOD

Dry distillation					Barometer, 587 mm		First drop: 51°C. (123°F.)	
Temperature, °C.	Percent cut	Sum, percent	Sp.gr. of cut	°A.P.I. of cut	Viscosity		Cloud test, °F.	Temperature, °F.
					At 100°F.	At 212°F.		
Up to 50								Up to 122
50-75	1.7	1.7	0.724	64.0				122 - 167
75-100	2.6	4.3	.752	56.7				167 - 212
100-125	5.1	9.4	.771	52.0				212 - 257
125-150	5.2	14.6	.800	45.4				257 - 302
150-175	5.2	19.8	.822	40.6				302 - 347
175-200	3.8	23.6	.835	38.0				347 - 392
200-225	4.1	27.7	.841	36.8				392 - 437
225-250	6.0	33.7	.844	36.2				437 - 482
250-275	8.2	41.9	.846	35.8				482 - 527

Vacuum distillation at 40 mm

Up to 200	3.5	3.5	0.843	36.4	44		40	Up to 392
200 - 225	8.9	12.4	.846	35.8	46		50	392 - 437
225 - 250	7.3	19.7	.856	33.8	53		80	437 - 482
250 - 275	7.8	27.5	.858	33.4	72*	37	100	482 - 527
275 - 300	9.0	36.5	.862	31.9	110*	40	115	527 - 572

Carbon residue of residuum, 5.9 percent. Carbon residue of crude, 1.2 percent.

APPROXIMATE SUMMARY

	Percent	Sp.gr.	°A.P.I.	Viscosity
Light gasoline	4.3	0.741	59.4	
Total gasoline and haphtha	19.8	.785	48.7	
Kerosene distillate	.0			
Gas oil	35.4	.843	36.4	
Nonviscous lubricating distillate	16.6	.853-.861	34.4-32.8	50 - 100
Medium lubricating distillate	6.6	.861-.864	32.8-32.3	100 - 200
Viscous lubricating distillate	.0			Above 200
Residuum	18.9	.941	18.9	
Distillation loss	2.7			

*Estimated from A. S. T. M. viscosity temperature chart.

REPORT OF CRUDE PETROLEUM ANALYSIS

Sample 36L-9

Cash No. 2
3,503 feet
Ohio Oil Co.

Lance Creek field
Second Dakota sand
Sampled, April 1936 NW SW Sec. 4, T. 35 N., R. 65 W

Wyoming
Niobrara County

GENERAL CHARACTERISTICS

Specific gravity, 0.836
Percent sulphur, less than 0.1
Saybolt Universal viscosity at 100°F., 42 sec.

A.P.I. gravity, 37.8°
Pour point, 40° F.
Color, blue-green

DISTILLATION, BUREAU OF MINES, HEMPEL METHOD

Dry distillation		Barometer, 586 mm		First drop: 52°C. (126°F.)			
Temperature, °C.	Per- cent cut	Sum, per- cent	Sp.gr. of cut	°A.P.I. of cut	Viscosity at 100°F.	Cloud test, °F.	Temperature, °F.
Up to 50							Up to 122
50-75	1.4	1.4	0.712	67.2			122 - 167
75-100	1.6	3.0	.723	62.9			167 - 212
100-125	6.0	9.0	.749	58.4			212 - 257
125-150	7.3	16.3	.772	51.8			257 - 302
150-175	6.1	22.4	.790	47.6			302 - 347
175-200	7.4	29.8	.805	44.3			347 - 392
200-225	5.3	35.1	.819	41.3			392 - 437
225-250	6.4	41.5	.831	38.8			437 - 482
250-275	7.7	49.2	.840	37.0			482 - 527

Vacuum distillation at 40 mm

Up to 200	0.8	0.8	0.847	35.6	45	20	Up to 392
200 - 225	8.4	9.2	.850	35.0	48	30	392 - 437
225 - 250	8.9	18.1	.857	33.6	56	50	437 - 482
250 - 275	7.4	25.5	.865	32.1	83	70	482 - 527
275 - 300	7.1	32.6	.872	30.8	127	90	527 - 572

Carbon residue of residuum, 1.4 percent. Carbon residue of crude, 0.3 percent.

APPROXIMATE SUMMARY

	Percent	Sp.gr.	°A.P.I.	Viscosity
Light gasoline	3.0	0.721	64.7	
Total gasoline and naphtha	29.3	.777	50.6	
Kerosene distillate	5.3	.819	41.3	
Gas oil	21.2	.840	37.0	
Nonviscous lubricating distillate	17.6	.852-.868	34.6-31.5	50 - 100
Medium lubricating distillate	7.9	.863-.875	31.5-30.2	100 - 200
Viscous lubricating distillate	.0	--	--	Above 200
Residuum	18.2	.904	25.0	
Distillation loss	.0			

REPORT OF CRUDE PETROLEUM ANALYSIS

Sample 36L-7

Richards & Comstock No. 2 Lance Creek field Wyoming
 3,624 feet Upper Sundance Niobrara County
 Ohio Oil Co. Sampled, April 1936 NE, SW sec. 32, T. 36 N., R. 65 W

GENERAL CHARACTERISTICS

Specific gravity, 0.812 A.P.I. gravity, 42.8°
 Percent sulphur, less than 0.1 Pour point, 10°F.
 Saybolt Universal viscosity at 70°F., 46 sec. Color, black

DISTILLATION, BUREAU OF MINES, HEMPEL METHOD

Dry distillation			Barometer, 592 mm		First drop: 29°C. (84°F.)		
Temperature, °C.	Per- cent cut	Sum, per- cent	Sp.gr. of cut	°A.P.I. of cut	Viscosity at 100°F.	Cloud test, °F.	Temperature, °F.
Up to 50	4.9	4.9	0.645	87.9			Up to 122
50- 75	3.4	8.3	.675	78.1			122 - 167
75-100	5.7	14.0	.711	67.5			167 - 212
100-125	6.3	20.3	.734	61.3			212 - 257
125-150	6.2	26.5	.752	56.7			257 - 302
150-175	5.3	31.8	.769	52.5			302 - 347
175-200	4.4	36.2	.782	49.4			347 - 392
200-225	5.2	41.4	.797	46.0			392 - 437
225-250	5.1	46.5	.809	43.4			437 - 482
250-275	8.2	54.7	.823	40.4			482 - 527

Vacuum distillation at 40 mm							
Up to 200	1.7	1.7	0.836	37.8	42	35	Up to 392
200 - 225	6.0	7.7	.841	36.8	45	45	392 - 437
225 - 250	5.6	13.3	.854	34.2	56	60	437 - 482
250 - 275	4.7	18.0	.865	32.1	72	80	482 - 527
275 - 300	5.8	23.8	.876	30.0	129	100	527 - 572

Carbon residue of residuum, 10.6 percent. Carbon residue of crude, 2.2 percent.

APPROXIMATE SUMMARY

	Percent	Sp.gr.	°A.P.I.	Viscosity
Light gasoline	14.0	0.679	76.9	
Total gasoline and naphtha	36.2	.727	63.1	
Kerosene distillate	18.5	.812	42.8	
Gas oil	7.4	.840	37.0	
Nonviscous lubricating distillate	10.9	.847-.870	35.6-31.1	50 - 100
Medium lubricating distillate	5.5	.870-.882	31.1-28.9	100 - 200
Viscous lubricating distillate	.0			Above 200
Residuum	17.9	.952	17.1	
Distillation loss	3.6			

REPORT OF CRUDE PETROLEUM ANALYSIS

Sample 36L-8

Lamb No. 7
3,880 feet
Ohio Oil Co.

Lance Creek field
Basal Sundance
Sampled, April 1936 NE, NE Sec. 5, T. 35 N., R. 65 W

Wyoming
Niobrara County

GENERAL CHARACTERISTICS

Specific gravity, 0.782
Percent sulphur, less than 0.1
Saybolt Universal viscosity at 70°F., 34 sec.

A.P.I. gravity, 49.40
Pour point, below 50°F.
Color, green

DISTILLATION, BUREAU OF MINES, HEMPEL METHOD

Dry distillation Barometer, 593 mm First drop: 26°C. (79°F.)

Temperature, °C.	Per- cent cut	Sum, per- cent	Sp.gr. of cut	°A.P.I. of cut	Viscosity at 100°F.	Cloud test, °F.	Temperature, °F.
Up to 50	4.5	4.5	0.653	85.2			Up to 122
50- 75	6.3	10.3	.663	81.9			122 - 167
75-100	8.3	19.1	.703	69.8			167 - 212
100-125	8.5	27.6	.733	61.5			212 - 257
125-150	6.9	34.5	.754	56.2			257 - 302
150-175	7.1	41.6	.772	51.8			302 - 347
175-200	4.6	46.2	.783	49.2			347 - 392
200-225	5.4	51.6	.796	46.3			392 - 437
225-250	5.5	57.1	.810	43.2			437 - 482
250-275	6.7	63.8	.824	40.2			482 - 527

Vacuum distillation at 40 mm

Up to 200	3.3	3.3	0.834	38.2	42	30	Up to 392
200 - 225	5.3	8.6	.843	36.4	46	45	392 - 437
225 - 250	4.6	13.2	.854	34.2	57	65	437 - 482
250 - 275	3.7	16.9	.864	32.3	78	80	482 - 527
275 - 300	6.2	23.1	.877	29.8	142	95	527 - 572

Carbon residue of residuum, 4.2 percent. Carbon residue of crude, 0.4 percent.

APPROXIMATE SUMMARY

	Percent	Sp.gr.	°A.P.I.	Viscosity
Light gasoline	19.1	0.678	77.2	
Total gasoline and naphtha	46.2	.724	63.9	
Kerosene distillate	17.6	.811	43.0	
Gas oil	7.8	.839	37.2	
Nonviscous lubricating distillate	8.9	.847-.868	35.6-31.5	50 - 100
Medium lubricating distillate	6.4	.868-.885	31.5-28.4	100 - 200
Viscous lubricating distillate	.0			Above 200
Residuum	8.8	.928	20.9	
Distillation loss	4.3			

REPORT OF CRUDE PETROLEUM ANALYSIS

Sample 37L-11

Composite of 5 wells Medicine Bow field
 5,500-5,520 feet First and second
 Ohio Oil Co. and California Co. Sundance sands
 Sampled, May 1937

Wyoming
 Carbon County
 T. 21 N., R. 79 W

GENERAL CHARACTERISTICS

Specific gravity, 0.726 A.P.I. gravity, 63.40
 Percent sulphur, less than 0.1 Pour point, below 5°F.
 Saybolt Universal viscosity at 70°F., 30 sec. Color, orange-yellow

DISTILLATION, BUREAU OF MINES, HEMPEL METHOD

Dry distillation		Barometer, 585 mm			First drop: 23°C. (73°F.)		
Temperature, °C.	Per- cent cut	Sum, per- cent	Sp.gr. of cut	°A.P.I. of cut	Viscosity at 100°F.	Cloud test, °F.	Temperature, °F.
Up to 50	14.7	14.7	0.638	90.3			Up to 122
50 - 75	10.4	25.1	.673	78.7			122 - 167
75 - 100	13.2	38.3	.714	66.7			167 - 212
100 - 125	11.9	50.2	.738	60.2			212 - 257
125 - 150	8.5	58.7	.761	54.4			257 - 302
150 - 175	7.3	66.0	.777	50.6			302 - 347
175 - 200	5.2	71.2	.791	47.4			347 - 392
200 - 225	4.7	75.9	.803	44.7			392 - 437
225 - 250	3.7	79.6	.816	41.9			437 - 482
250 - 275	4.1	83.7	.825	40.0			482 - 527

Vacuum distillation at 40 mm

Up to 200	0.7	0.7	0.847	35.6	44	35	Up to 392
200 - 225	1.7	2.4	.848	35.4	47	40	392 - 437
225 - 250	1.4	3.8	.856	33.8	57	55	437 - 482
250 - 275	.3	4.1	.859	33.2	64	70	482 - 527
275 - 300	1.6	5.7	.877	29.8	101	85	527 - 572

Carbon residue of residuum, 4.1 percent. Carbon residue of crude, less than 0.1 percent.

APPROXIMATE SUMMARY

	Percent	Sp.gr.	°A.P.I.	Viscosity
Light gasoline	38.3	0.674	78.4	
Total gasoline and naphtha	71.2	.714	66.7	
Kerosene distillate	12.5	.814	42.3	
Gas oil	1.8	.848	35.4	
Nonviscous lubricating distillate	3.1	.850-.877	35.0-29.8	50 - 100
Medium lubricating distillate	.8	.877-.891	29.8-27.3	100 - 200
Viscous lubricating distillate	--	--	--	Above 200
Residuum	1.0	.938	19.4	
Distillation loss	9.6			

REPORT OF CRUDE PETROLEUM ANALYSIS

Sample 36L-10

Gjullin No. 6
1,968 feet
Stanolind

Pondera field
Madison limestone
Sampled, May 1936 NW Sec. 15, T. 27 N., R. 4 W

Montana
Fondera County

GENERAL CHARACTERISTICS

Specific gravity, 0.866

A.P.I. gravity, 31.90

Percent sulphur, 1.59

Pour point, below 5°F.

Saybolt Universal viscosity at 100°F., 52 sec.

Color, black

DISTILLATION, BUREAU OF MINES, HEMPEL METHOD

Dry distillation		Barometer, 592 mm		First drop: 30°C. (87°F.)			
Temperature, °C.	Per- cent cut	Sum, per- cent	Sp.gr. of cut	°A.P.I. of cut	Viscosity at 100°F.	Cloud test, °F.	Temperature, °F.
Up to 50	3.2	3.2	0.667	80.6			Up to 122
50 - 75	6.7	9.9	.681	76.3			122 - 167
75 - 100	2.9	12.8	.720	65.0			167 - 212
100 - 125	2.6	15.4	.739	60.0			212 - 257
125 - 150	5.6	21.0	.759	54.9			257 - 302
150 - 175	5.2	26.2	.781	49.7			302 - 347
175 - 200	4.7	30.9	.797	46.0			347 - 392
200 - 225	4.8	35.7	.811	43.0			392 - 437
225 - 250	5.2	40.9	.830	39.0			437 - 482
250 - 275	6.6	47.5	.850	35.0			482 - 527

Vacuum distillation at 40 mm

Up to 200	0.3	0.3)	--	--			Up to 392
200 - 225	7.7	8.0)	0.876	30.0	47	35	392 - 437
225 - 250	6.0	14.0	.889	27.7	59	50	437 - 482
250 - 275	7.4	21.4	.908	24.3	96	75	482 - 527
275 - 300	4.6	26.0	.922	22.0	233	95	527 - 572

Carbon residue of residuum, 15.3 percent. Carbon residue of crude, 3.6 percent.

APPROXIMATE SUMMARY

	Percent	Sp.gr.	°A.P.I.	Viscosity
Light gasoline	12.8	0.687	74.5	
Total gasoline and naphtha	30.3	.736	60.7	
Kerosene distillate	4.8	.811	43.0	
Gas oil	17.7	.852	34.6	
Nonviscous lubricating distillate	12.0	.379-.909	29.5-24.2	50 - 100
Medium lubricating distillate	4.4	.909-.919	24.2-22.5	100 - 200
Viscous lubricating distillate	3.7	.919-.927	22.5-21.1	Above 200
Residuum	23.1	1.009	8.8	
Distillation loss	3.3	--	--	

REPORT OF CRUDE PETROLEUM ANALYSIS

Sample 36L-18

Carl Allen Well No. 1 Powder Wash field Colorado
 Depth, 5,032 feet Lower Wasatch (Tertiary) Moffat County
 Mountain Fuel Supply Co. Sampled, November 1936 NE SW SE Sec. 32, T. 12 N., R. 97 W

GENERAL CHARACTERISTICS

Specific gravity, 0.850 A.P.I. gravity, 35.00
 Percent sulphur, 0.11 Pour point, 85°F.
 Saybolt Universal viscosity at 100°F., 48 sec. Color, green

DISTILLATION, BUREAU OF MINES, HEMPEL METHOD

Dry distillation Barometer, 586 mm First drop: 91°C. (196°F.)

Temperature, °C.	Per-cent cut	Sum, per-cent	Sp.gr. of cut	°A.P.I. of cut	Viscosity		Cloud test, °F.	Temperature, °F.
					At 100°F.	At 212°F.		
Up to 50								Up to 122
50- 75								122 - 167
75-100	1.3	1.3	0.773	51.6				167 - 212
100-125	4.2	5.5	.779	50.1				212 - 257
125-150	7.0	12.5	.790	47.6				257 - 302
150-175	4.6	17.1	.797	46.0				302 - 347
175-200	5.0	22.1	.804	44.5				347 - 392
200-225	6.1	28.2	.814	42.3				392 - 437
225-250	7.6	35.8	.831	38.8				437 - 482
250-275	8.9	44.7	.840	37.0				482 - 527

Vacuum distillation at 40 mm

Up to 200	3.3	3.3	0.845	36.0	43		45	Up to 392
200 - 225	8.0	11.3	.849	35.2	46		60	392 - 437
225 - 250	7.1	18.4	.856	33.8	56		80	437 - 482
250 - 275	8.0	26.4	.861	32.8	74*	37	100	482 - 527
275 - 300	10.2	36.6	.863	32.5	103*	40	120	527 - 572

Carbon residue of residuum, 3.3 percent. Carbon residue of crude, 0.7 percent.

APPROXIMATE SUMMARY

	Percent	Sp.gr.	°A.P.I.	Viscosity
Light gasoline	1.3	0.773	51.6	
Total gasoline and naphtha	22.1	.792	47.2	
Kerosene distillate	6.1	.814	42.3	
Gas oil	26.5	.840	37.0	
Nonviscous lubricating distillate	20.6	.852-.863	34.6-32.5	50 - 100
Medium lubricating distillate	6.0	.863-.864	32.5-32.3	100 - 200
Viscous lubricating distillate	.0			Above 200
Residuum	17.4	.936	19.7	
Distillation loss	1.3			

*Viscosity at 100°F. estimated from ASTM vis.-temp. chart.

REPORT OF CRUDE PETROLEUM ANALYSIS

Sample 36L-1¹⁴

Wilson No. 1
3,401 feet
California Co.

Quealy Dome field
Dakota sand
Sampled, August 1936 NW, SW, SW Sec. 18, T. 17 N., R. 76 W

Wyoming
Albany County

GENERAL CHARACTERISTICS

Specific gravity, 0.859
Percent sulphur, 0.27
Saybolt Universal viscosity at 100°F., 54.0 sec.

A.P.I. gravity, 33.2°
Pour point, 15° F.
Color, greenish brown

DISTILLATION, BUREAU OF MINES, HEMPEL METHOD

Dry distillation Barometer, 588 mm First drop: 23°C. (73°F.)

Temperature, °C.	Percent cut	Sum, percent	Sp.gr. of cut	°A.P.I. of cut	Viscosity		Cloud test, °F.	Temperature, °F.
					At 100°F.	At 212°F.		
Up to 50	1.9	1.9	0.669	80.0				Up to 122
50-75	2.1	4.0	.679	76.9				122 - 167
75-100	4.2	8.2	.715	66.4				167 - 212
100-125	4.9	13.1	.740	59.7				212 - 257
125-150	4.2	17.3	.762	54.2				257 - 302
150-175	3.7	21.0	.781	49.7				302 - 347
175-200	3.8	24.8	.798	45.8				347 - 392
200-225	4.4	29.2	.813	42.5				392 - 437
225-250	4.8	34.0	.826	39.8				437 - 482
250-275	6.2	40.2	.833	38.4				482 - 527

Vacuum distillation at 40 mm

Up to 200	2.3	2.3	0.844	36.2	41		30	Up to 392
200 - 225	6.0	3.3	.853	34.4	47		45	392 - 437
225 - 250	8.1	16.4	.866	31.9	58		65	437 - 482
250 - 275	6.7	23.1	.879	29.5	100		85	482 - 527
275 - 300	8.4	31.5	.890	26.9	138*	48	105	527 - 572

Carbon residue of residuum, 8.3 percent. Carbon residue of crude, 2.6 percent.

APPROXIMATE SUMMARY

	Percent	Sp.gr.	°A.P.I.	Viscosity
Light gasoline	8.2	0.696	71.8	
Total gasoline and naphtha	24.8	.744	58.7	
Kerosene distillate	4.4	.813	42.5	
Gas oil	18.3	.838	37.4	
Nonviscous lubricating distillate	12.5	.857-.879	33.6-29.5	50 - 100
Medium lubricating distillate	8.5	.879-.891	29.5-27.3	100 - 200
Viscous lubricating distillate	3.2	.891-.896	27.3-26.4	Above 200
Residuum	28.1	.965	15.1	
Distillation loss	.2			

*By extrapolation of temperature-viscosity curve.

REPORT OF CRUDE PETROLEUM ANALYSIS

Sample 36L-16

Diamond Cattle No. 5
3,188 feet
Ohio Oil Co.

Rock Creek field
Sundance sand
Sampled, September 1936 NW, SW, NW Sec. 35, T. 20 N., R. 78 W

Wyoming
Carbon County

GENERAL CHARACTERISTICS

Specific gravity, 0.845

A.P.I. gravity, 36.0°

Percent sulphur, 0.16

Pour point, 20°F.

Saybolt Universal viscosity at 100°F., 43 sec.

Color, greenish brown

DISTILLATION, BUREAU OF MINES, HEMPEL METHOD

Dry distillation Barometer, 591 mm First drop: 24°C. (76°F.)

Temperature, °C.	Per- cent cut	Sum, per- cent	Sp.gr. of cut	°A.P.I. of cut	Viscosity at 100°F.	Cloud test, °F.	Temperature, °F.
Up to 50	1.8	1.8	0.663	81.9			Up to 122
50 - 75	3.2	5.0	.677	77.5			122 - 167
75 - 100	6.0	11.0	.715	66.4			167 - 212
100 - 125	6.4	17.4	.741	59.4			212 - 257
125 - 150	5.3	22.7	.765	53.5			257 - 302
150 - 175	4.1	26.8	.784	49.0			302 - 347
175 - 200	4.6	31.4	.800	45.4			347 - 392
200 - 225	4.4	35.8	.816	41.9			392 - 437
225 - 250	5.4	41.2	.828	39.4			437 - 482
250 - 275	6.4	47.6	.840	37.0			482 - 527

Vacuum distillation at 40 mm

Up to 200	1.8	1.8	0.853	34.4	43	30	Up to 392
200 - 225	6.8	8.6	.858	33.4	48	45	392 - 437
225 - 250	4.8	13.4	.869	31.3	60	60	437 - 482
250 - 275	5.5	18.9	.879	29.5	83	80	482 - 527
275 - 300	7.7	26.6	.895	26.6	171	95	527 - 572

Carbon residue of residuum, 7.8 percent. Carbon residue of crude, 2.1 percent.

APPROXIMATE SUMMARY

	Percent	Sp.gr.	°A.P.I.	Viscosity
Light gasoline	11.0	0.696	71.8	
Total gasoline and naphtha	31.4	.743	58.9	
Kerosene distillate	4.4	.816	41.9	
Gas oil	18.0	.842	36.6	
Nonviscous lubricating distillate	11.2	.860-.882	33.0-28.9	50 - 100
Medium lubricating distillate	7.4	.882-.900	28.9-25.7	100 - 200
Viscous lubricating distillate	1.8	.900-.904	25.7-25.0	Above 200
Residuum	23.3	.960	15.9	
Distillation loss	2.0			

REPORT OF CRUDE PETROLEUM ANALYSIS

Sample 36L-6

Well No. 1 Waugh Dome field Wyoming
 3,807 feet Embar lime Hot Springs County
 California Exploration Co. Sampled, July 1936 NE $\frac{1}{4}$ Sec. 12, T. 44 N., R. 97 W.

GENERAL CHARACTERISTICS

Specific gravity, 0.888 A.P.I. gravity, 27.8°
 Percent sulphur, 1.77 Pour point, below 0°F.
 Saybolt Universal viscosity at 100°F., 77 sec. Color, black

DISTILLATION, BUREAU OF MINES, HEMPEL METHOD

Dry distillation		Barometer, 593 mm		First drop: 47°C. (116°F.)			
Temperature, °C.	Per- cent cut	Sum, per- cent	Sp.gr. of cut	°A.P.I. of cut	Viscosity at 100°F.	Cloud test, °F.	Temperature, °F.
Up to 50	0.3	0.3					Up to 122
50 - 75	1.7	2.0	0.680	76.6			122 - 167
75 - 100	2.3	4.3	.703	69.8			167 - 212
100 - 125	3.5	7.8	.726	63.4			212 - 257
125 - 150	3.6	11.4	.747	57.9			257 - 302
150 - 175	3.8	15.2	.771	52.0			302 - 347
175 - 200	4.7	19.9	.794	46.7			347 - 392
200 - 225	4.6	24.5	.813	42.5			392 - 437
225 - 250	5.7	30.2	.827	39.6			437 - 482
250 - 275	7.9	38.1	.841	36.8			482 - 527

Vacuum distillation at 40 mm

Up to 200	1.4	1.4	0.861	32.8	43	15	Up to 392
200 - 225	7.9	9.3	.870	31.1	49	25	392 - 437
225 - 250	6.8	16.1	.887	28.0	61	45	437 - 482
250 - 275	7.4	23.5	.906	24.7	97	65	482 - 527
275 - 300	6.9	30.4	.919	22.5	202	85	527 - 572

Carbon residue of residuum, 17.9 percent. Carbon residue of crude, 6.4 percent.

APPROXIMATE SUMMARY

	Percent	Sp.gr.	°A.P.I.	Viscosity
Light gasoline	4.3	0.692	73.0	
Total gasoline and naphtha	19.9	.747	57.9	
Kerosene distillate	4.6	.813	42.5	
Gas oil	19.8	.844	36.2	
Nonviscous lubricating distillate	13.8	.872-.906	30.8-24.7	50 - 100
Medium lubricating distillate	6.7	.906-.919	24.7-22.5	100 - 200
Viscous lubricating distillate	3.7	.919-.925	22.5-21.5	Above 200
Residuum	31.5	1.000	10.0	
Distillation loss	0.0			

REPORT OF CRUDE PETROLEUM ANALYSIS

Sample 37L-5

Wertz lease A and B, well No. 10 Wertz field Wyoming
 5,872 - 5,886 feet Tensleep sand Carbon County
 Sinclair-Wyoming Oil Co. Sampled, April 1937 Sec. 7, T. 26 N., R. 89 W

GENERAL CHARACTERISTICS

Specific gravity, 0.849 A.P.I. gravity, 35.2°
 Percent sulphur, 1.30 Pour point, below 5° F.
 Saybolt Universal viscosity at 100° F., 41 sec. Color, brownish-green

DISTILLATION, BUREAU OF MINES, HEMPEL METHOD

Dry distillation		Barometer, 588 mm			First drop: 27° C. (80° F.)		
Temperature, °C.	Per- cent cut	Sum, per- cent	Sp.gr. of cut	°A.P.I. of cut	Viscosity at 100° F.	Cloud test, °F.	Temperature, °F.
Up to 50	1.9	1.9	0.665	81.3			Up to 122
50 - 75	4.4	6.3	.666	81.0			122 - 167
75 - 100	3.7	10.0	.702	70.1			167 - 212
100 - 125	5.2	15.2	.731	62.1			212 - 257
125 - 150	5.5	20.7	.757	55.4			257 - 302
150 - 175	5.3	26.0	.778	50.4			302 - 347
175 - 200	4.8	30.8	.794	46.7			347 - 392
200 - 225	5.1	35.9	.809	43.4			392 - 437
225 - 250	5.8	41.7	.828	39.4			437 - 482
250 - 275	7.2	48.9	.848	35.4			482 - 527

Vacuum distillation at 40 mm

Up to 200	1.0	1.0	0.869	31.3	46	30	Up to 392
200 - 225	8.9	9.9	.876	30.0	50	35	392 - 437
225 - 250	5.5	15.4	.892	27.1	69	55	437 - 482
250 - 275	5.3	20.7	.905	24.8	105	70	482 - 527
275 - 300	6.8	27.5	.915	23.1	212	85	527 - 572

Carbon residue of residuum, 9.3 percent. Carbon residue of crude, 2.5 percent.

APPROXIMATE SUMMARY

	Percent	Sp.gr.	°A.P.I.	Viscosity
Light gasoline	10.0	0.679	76.9	
Total gasoline and naphtha	30.8	.737	60.5	
Kerosene distillate	5.1	.809	43.4	
Gas oil	18.5	.849	35.2	
Nonviscous lubricating distillate	11.9	.876-.903	30.0-25.2	50 - 100
Medium lubricating distillate	6.0	.903-.914	25.2-23.3	100 - 200
Viscous lubricating distillate	4.1	.914-.920	23.3-22.3	Above 200
Residuum	23.5	.972	14.1	
Distillation loss	.1			

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REPORT OF INVESTIGATIONS

PERMISSIBLE METHANE DETECTORS

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BY

L. C. ILSLEY AND A. B. HOOKER

REPORT OF INVESTIGATIONS

UNITED STATES DEPARTMENT OF THE INTERIOR - BUREAU OF MINES

PERMISSIBLE METHANE DETECTORS^{1/}
(Supplement to Bulletin 331)

By L. C. Ilsley^{2/} and A. B. Hooker^{3/}

The last previous Bureau of Mines publication relative to details of permissible methane detectors (Bulletin 331, published in 1930) describes detectors of four different types, namely flame, combustion, catalytic, and a combination Wheatstone-bridge and catalytic type in which catalytic action at the surface of a platinum filament when in methane causes a change in the temperature and resistance of the filament, thus unbalancing the bridge circuit and giving a reading proportional to the methane concentration. Three detectors of the latter type have been approved since 1930. A brief description of each follows:

U. C. C. Detector

Approval No. 802, covering the U. C. C. indicating detector, as described in Bulletin 331, was extended January 27, 1936, to cover important changes in design that permit checking the calibration of the detector at any time whether in gas or fresh air. Essential features of the detector are shown in figure 1.

The detector head, a safety compartment that houses the filament, is the part of the detector that is placed in the gassy mixture when readings are taken. It consists of an aluminum bonnet (a) fitted to an aluminum base (b) having an extension or handle (c). Within the bonnet are three concentric brass gauzes (d) with aluminum flange bases. Inside the inner gauze is the platinum filament (e). Contact to the filament is completed when pins (f), which are attached to the base of the middle gauze, are pushed downward on insulators (g) against the action of safety contact springs (h). Thus, the filament cartridge must first be assembled in the inner gauze and covered with the second or middle gauze before connection to the battery is completed. This insures at least double gauze protection of the lighted filament and permits replacement of a burned-out filament safely any place in the mine.

- 1/ The Bureau of Mines will welcome reprinting of this paper provided the following footnote acknowledgment is used: "Reprinted from Bureau of Mines Report of Investigations 3359."
- 2/ Electrical engineer, Bureau of Mines, Pittsburgh Experiment Station, Pittsburgh, Pa.
- 3/ Associate electrical engineer, Bureau of Mines, Pittsburgh Experiment Station, Pittsburgh, Pa.

The handle is provided with ribs (i), placed in a definite relation to the axis of the filament as a guide to holding the detector head in the correct position when testing. The complete head weighs about 2 pounds and is carried in the hand; provision is made for hanging it, when not in use, on the user's belt or clothing.

The meter and bridge compartment contains the several bridge resistances and the meter. The general arrangement of the compartment is as follows: The meter (j) is mounted at the top of the compartment and is protected by a metal guard (k) having an opening through which the calibrated scale may be seen. A combination battery and meter switch is operated by a knob (s) in a recess at the bottom of the compartment. Another knob (m) allows adjustment of resistance to balance the Wheatstone bridge circuit when making zero meter-scale adjustment. The ears (n) fastened to the side of the compartment are for attaching the carrying straps.

The compartment weighs 2-1/2 pounds and is strapped to the observer in such a position that the meter scale is readily visible.

Cord (o) connects the detector to a lead-acid battery, which also supplies current for a cap lamp type lighting attachment.

The electrical circuit of this detector is of the Wheatstone bridge type. With the switch (S) turned to the right in the reading position and the filament at its normal temperature, the current flow from the battery through the bridge from points 1 to 4 is adjusted by resistance knob (m) to such a value that no current flows from 2 to 3 when the meter pointer is at zero scale reading. When the detector head is in gas, the filament increases in temperature and resistance, forcing a part of the current through the path 3, 2, 4, which includes the meter. This gives a scale reading proportional to the percentage of methane around the filament.

Changes in the zero scale setting occur because of changes in terminal voltage as the battery discharges. The present model of the U. C. C. detector permits checking the original setting at any time. The checking is accomplished as follows:

After closing switch (S) in the reading position and adjusting to zero reading in air, throw the switch to the check position and move the index pointer (x) to mark the position of the needle. This is the check reading position with which the needle must coincide in subsequent checkings.

The complete detector, including the meter and bridge compartment, head, battery, and lighting attachment weighs 12-1/4 pounds.

M. S. A. Type AP-6 Detector (Approval No. 804)

This detector also operates on the Wheatstone-bridge principle, its electrical circuit being a modified Wheatstone bridge called an auto-potentiometer.

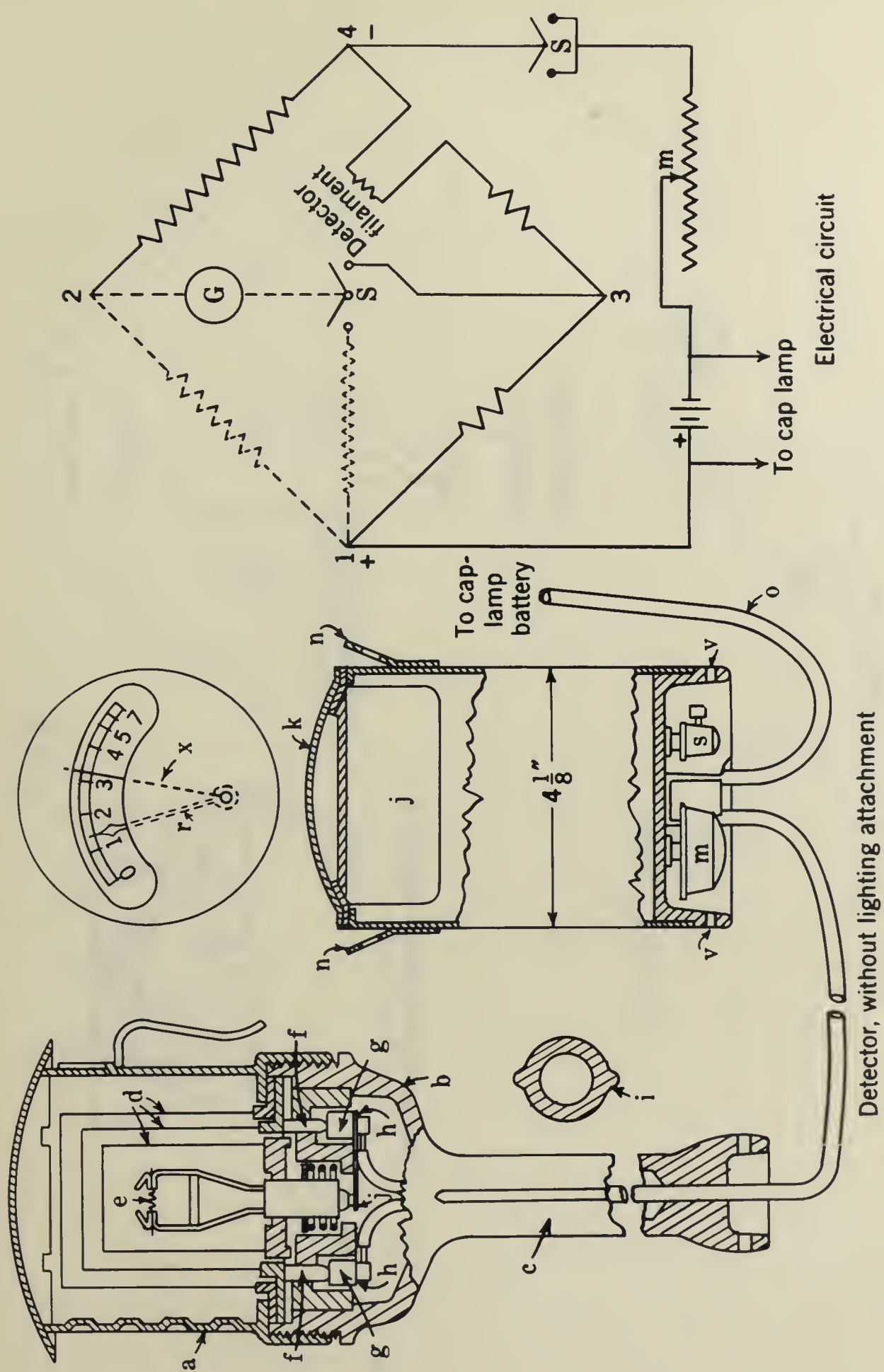


Figure 1.—U. C. C. permissible methane detector.

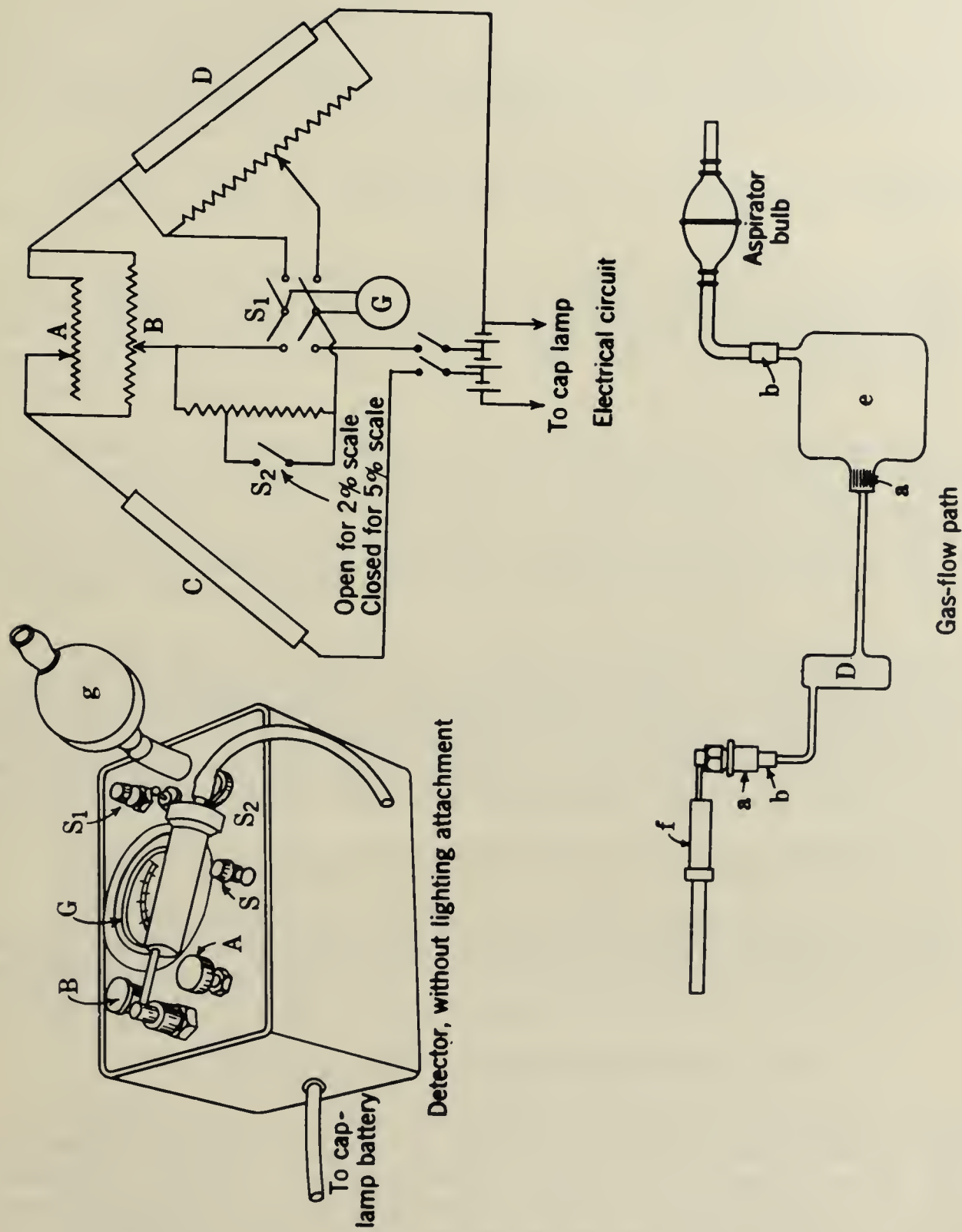


Figure 2.—M. S. A. type AP-6 permissible methane detector.

The chief features of the detector are: (1) It has two filaments; one, called the detector filament, that is in the flow line through which the mixture to be sampled is circulated by means of an aspirator bulb, the other a compensator filament sealed in fresh air. (2) A means of checking the zero setting by waiting until the gas in the aspirated sample around the detector filament is burned out. (3) Two scales, 0-5 and 0-2, respectively, giving greater convenience and accuracy of determination. (4) A dryer in the sampling line to insure a more uniform sample and greater accuracy of reading, especially with low-percentage concentrations.

Figure 2 shows the general appearance of the detector together with its electrical circuit and gas flow path.

In operating the AP-6 detector, the check calibration is made by means of rheostat (A) with switch (S_1) in the check position, to the right, and the aspirator bulb (g) not operated. Zero scale calibration is made in fresh air by means of rheostat (B) with switch (S_1) in the reading position and while aspirating.

Aspirating reduces the pressure in tank (e) and produces an approximately uniform flow of the sample past the detector filament. The rate of flow is limited by the resistance of the path, including flow orifices (b), drying tube (f), and flame arrestors (a), one of which is placed on each side of the filament compartment to insure that flame does not pass to the outside to cause possible ignition of external mixtures.

This detector is approved in connection with three Edison permissible electric cap lamps - the model H lamp with a 2-cell battery; the model J lamp with a 2-cell battery, and the model K lamp with a 3-cell battery, as shown in figure 2. The total weights of the detector with the above lamps are 16-7/8, 15-3/4, and 17-1/5 pounds, respectively, the weight of the detector with carrying straps but without the lamp being 11-3/4 pounds.

M. S. A. Type AP-5 Detector Dry-Cell Type (Approval No. 803)

The type AP-5 detector is the same as the type AP-6, with which it was tested and approved, except that it has no lighting attachment and has a battery of two no. 6 dry cells inside the detector housing; this detector is therefore self-contained. It weighs approximately 16 pounds.

M. S. A. Type W3 Detector Approval No. 805

This detector, of later design than the type AP detectors, operates on the true Wheatstone bridge principle. Like the AP detectors, it has two filaments - detector and compensator - but both filaments are in the gas-flow circuit and are thus subjected to the same gas mixture. The filaments are electrically connected in series across the whole battery. Other improvements include a provision for quickly checking the calibration at any time, the elimination of the expansion chamber and dryer, and an appreciable decrease in the size and weight of the detector.

Figure 3 shows the detector, together with the electrical circuit and the gas flow path.

Opening the detector cover automatically closes switch (S) connecting the battery to the bridge circuit.

Approximate adjustment of the detector is made by depressing switch (S_1) to the check position and obtaining the proper check reading by adjusting (A). Then with switch (S_1) released or in the reading position, adjustment to zero scale reading is made by means of (B). This adjustment relative to both checking and reading positions in air balances the bridge circuit so that no current is flowing through the meter (G).

When mine atmosphere containing methane is aspirated through the flow circuit, the activated detector filament (D) increases in temperature and resistance proportional to the methane content, thereby unbalancing the bridge circuit and giving a meter scale reading in percent methane.

After the initial setting in fresh air, the W-8 detector can be checked at any time, whether methane is present or not, and corrections made by rheostat (A) for normal changes in the battery voltage.

The rate of gas flow through the filament compartments is limited by the combined effect of flashback arresters (a), the flow orifices (b), the bypass inlet (e), cleaning screen (f), and calibration holes through the filament cartridges, not shown.

The W-8 detector is approved for use with the Edison model K permissible cap lamp. The total weight of the detector with lamp is 11-7/8 pounds; the weight of the detector only, but with carrying straps, is 6-1/2 pounds.

CONCLUSIONS

The comparatively good record in United States coal mines relative to freedom from mine disasters during recent years has undoubtedly followed greater attention to mine ventilation. Indicating type detectors that permit more accurate determinations of methane than are obtainable with flame safety lamps are a great aid in controlling ventilation.

All three detectors described in this pamphlet represent improvements over former designs that should insure even more satisfactory performance. However, in order to obtain satisfactory results care must be used in the proper maintenance of the detectors. Those using them need to be thoroughly familiar with their use and have the skill necessary in handling apparatus of this kind.

With the U. C. C. and the M. S. A. type W-8 detectors it is possible and desirable to check the setting before each methane determination and thus assure reasonably accurate results throughout the working voltage range of the battery.

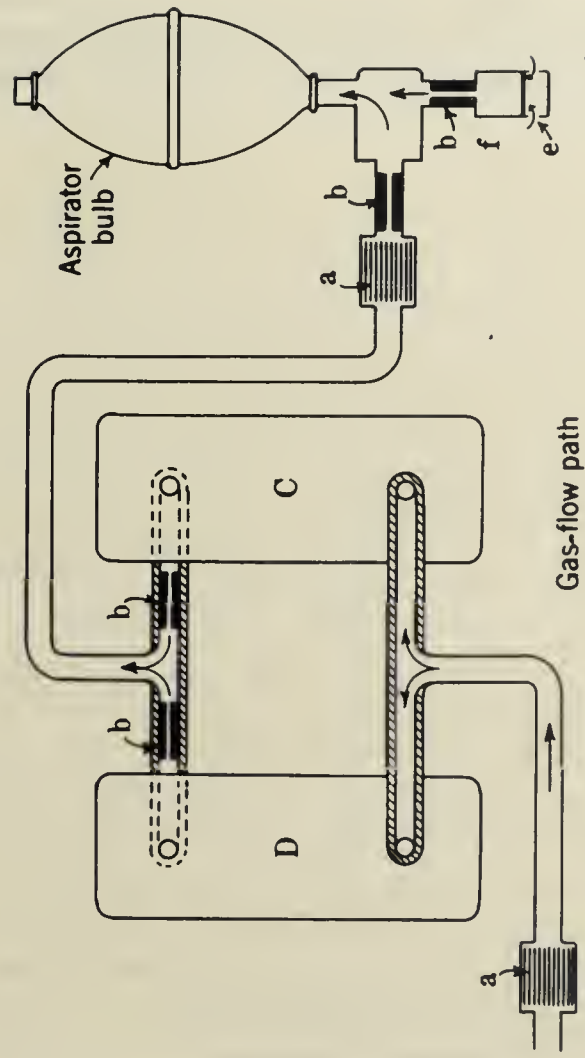
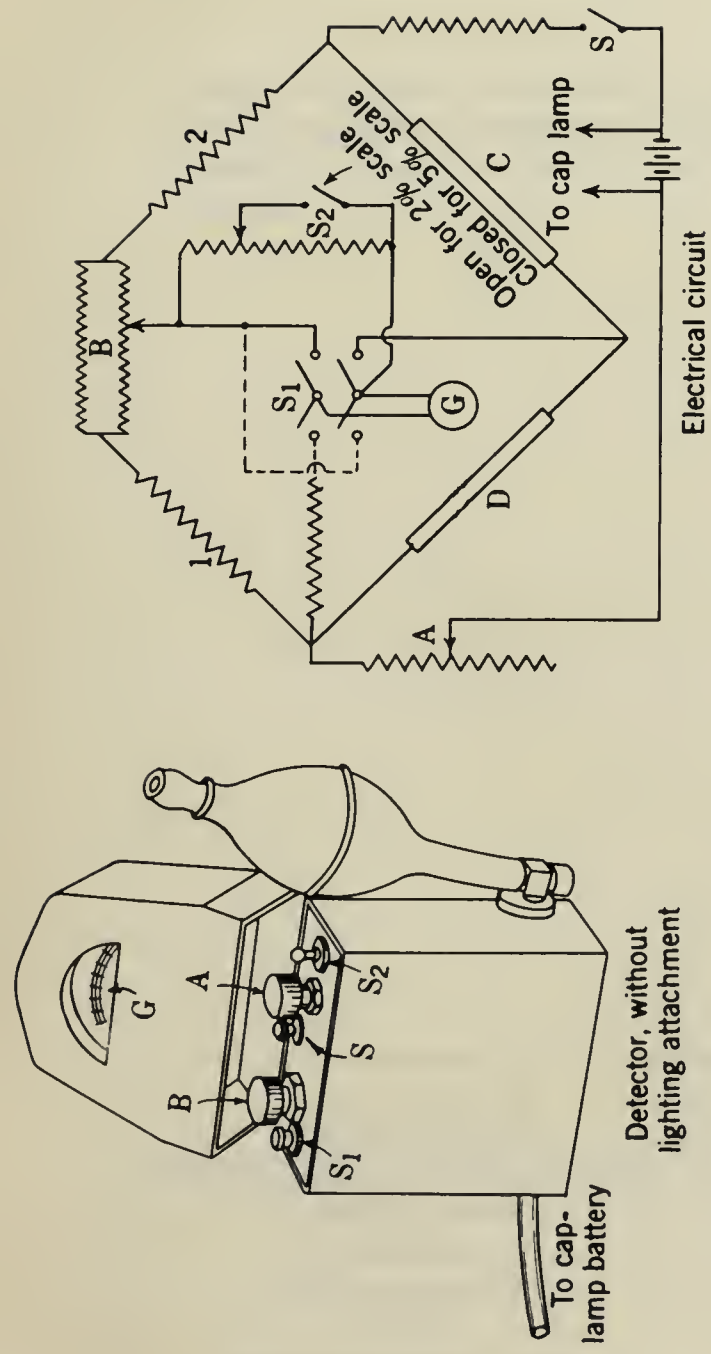


Figure 3.—M. S. A. type W-8 permissible methane detector.

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REPORT OF INVESTIGATIONS

ACTIVE LIST OF PERMISSIBLE EXPLOSIVES AND BLASTING
DEVICES APPROVED PRIOR TO JUNE 30, 1937



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REPORT OF INVESTIGATIONS

UNITED STATES DEPARTMENT OF THE INTERIOR - BUREAU OF MINES

ACTIVE LIST OF PERMISSIBLE EXPLOSIVES AND BLASTING DEVICES
APPROVED PRIOR TO JUNE 30, 1937^{1/ 2/}

PERMISSIBLE EXPLOSIVES TESTED UNDER SCHEDULE 17C

An active list of permissible explosives was published in December 1936 as part of Bureau of Mines Report of Investigations 3324, under the title "Active List of Permissible Explosives and Blasting Devices Approved Prior to July 31, 1936." This list included 170 brand names.

Changes in the list that have been authorized to June 30, 1937 placed 35 additional brands of permissible explosives on the active permissible list as a result of tests, changes in brand names, or transfer from the inactive list. Ten permissible explosives were transferred from the active to the inactive list. The present list contains 196 brand names, one explosive having been placed on the list subsequent to June 30, 1937. The brand names of the permissible explosives placed on the active list are as follows:

Anthracite 1	By change in name.
Anthracite 2	Do.
Anthracite 3	Do.
Anthracite 4	Do.
Big Red No. 7-C	By test.
Black Diamond Special C ^{3/}	Do.
Burcoal 2, L. F.	By transfer from inactive.
Burcoal 4, L. F.	Do.
Burcoal 5, L. F.	By change in name.
Burton 1, L. F.	By transfer from inactive.
Burton 10-A	By test.
Coalite LS, L. F.	Do.
Genite 2	By change in name.
Genite 4	Do.
Genite 10-A	Do.
King No. 5-A	Do.
King No. 7	Do.
King No. 7-A	Do.

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2/ Prepared by J. E. Tiffany, Explosives Division, Bureau of Mines Experiment Station, Pittsburgh, Pa.

3/ Placed on list July 1937.

King No. 8	By change in name.
King No. 9	Do.
King No. 11	Do.
King No. 11-A	Do.
King No. 12	Do.
King No. 12-A	Do.
King No. 15	Do.
King A	Do.
King G	Do.
King Nu-Gel No. 1	Do.
King Nu-Gel No. 2	Do.
King Nu-Gel No. 4	Do.
King Special A	Do.
National A	By test.
National B	Do.
National C	Do.
National D	Do.
Permigel 1, L. F.	By transfer from inactive.

The permissible explosives transferred from the active to the inactive list are as follows:

Austin Red Diamond No. 12-A, L. F.	Monobel No. 1, L. F.
Coalite C, L. F.	Monobel 8, L. F. -WR
Duobel No. 2, L. F.	Monobel No. 11, L. F.
Duobel L. F.	Red H No. 4-A, L. F.
Mine-ite No. 6-D	Tri-State Special No. 1

In testing all explosives now on the active permissible list a no. 6 electric detonator was used, and no detonator of less efficiency should be used in firing permissible explosives in coal mines.

CLASSIFICATION ON BASIS OF POISONOUS GASES

All permissible explosives when detonated emit some poisonous gases and a much larger volume of nonpoisonous gases. In order that the poisonous gases may not under normal conditions become a menace to the lives or health of miners, no explosive is now or can become permissible if it evolve upon detonation more than 158 liters (5-1/2 cubic feet) of poisonous gases per 1-1/2 pound charge, as determined by tests in the Bichel pressure gage.

The classification on the basis of the volume of poisonous gases produced by 680 grams (1-1/2 pounds) of explosive is as follows: Class A, not more than 53 liters; class B, between 53 and 106 liters; and class C, between 106 and 158 liters.

Field tests were made with a 1-1/2 pound charge of a permissible explosive which produced, in the Bichel gage, the maximum allowable quantity of poisonous gases (158 liters per 1-1/2 pounds); these tests indicated that in a narrow entry, without artificial ventilation, 0.18 percent of carbon monox-

ide (the only poisonous gas present) was produced, as shown by analysis of an air sample taken 2 minutes after the shot. Another sample of the air taken 2 minutes later contained 0.08 percent of carbon monoxide. Therefore, under no conditions should miners or shot firers return to the face until at least 5 minutes after a shot.

The complete active list of permissible explosives is shown in the following table:

NONGELATINOUS PERMISSIBLE EXPLOSIVES

Brand	Class designation basis, of volume of poisonous gases	Weight of 1-1/4-inch cartridge	Small-diameter	Rate of detonation in 1-1/4-inch-diameter cartridge						Manufacturer
				Feet per second		Meters per second				
				Lower limit	Rate terminated	Upper limit	Lower limit	Rate terminated	Upper limit	
1	2	3	4	5						6
		Grams	Inches							
Anthracite 12/	A	160	1	7,200	8,500	9,800	2,200	2,600	3,000	Independent
Anthracite 23/	A	147	1	8,900	10,500	12,100	2,700	3,200	3,700	Do.
Anthracite 34/	B	100	1	7,500	8,900	10,200	2,300	2,700	3,100	Do.
Anthracite 45/	A	162	1	5,900	6,900	7,900	1,800	2,100	2,400	Do.
Apache Coal Powder A	B	137	1-1/4	10,000	11,300	13,600	3,100	3,600	4,100	Apache
Apache Coal Powder B	B	158	1-1/4	7,000	8,200	9,400	2,100	2,500	2,900	Do.
Apache Coal Powder H, L. F.	A	164	1-1/8	9,200	10,800	12,400	2,800	3,300	3,800	Do.
Apcol No. 1, L. F.	A	175	7/8	6,400	7,500	8,700	2,000	2,300	2,600	Atlas-Giant
Austin Red Diamond No. 1, L. F.	B	173	1	8,900	10,500	12,100	2,700	3,200	3,700	Austin
Austin Red Diamond No. 4, L. F.	A	191	1	8,400	9,800	11,300	2,600	3,000	3,400	Do.
Austin Red Diamond No. 5, L. F.	A	161	1	6,700	7,900	9,100	2,000	2,400	2,800	Do.
Austin Red Diamond No. 9-B, L.F.	B	133	7/8	6,400	7,500	8,700	2,000	2,300	2,600	Do.
Austin Red Diamond No. 9-C, L.F.	B	144	7/8	7,200	8,500	9,800	2,200	2,600	3,000	Do.
Austin Red Diamond No. 10-A, L.F.	B	123	7/8	7,000	8,200	9,400	2,100	2,500	2,900	Do.
Austin Red Diamond No. 11, L.F.	B	100	7/8	5,000	5,900	6,800	1,500	1,800	2,100	Do.
Big Red No. 7	A	186	7/8	8,400	9,800	11,300	2,600	3,000	3,400	Equitable

1/ The classification on the basis of the volume of poisonous gases produced is given on page 2.

2/ Same as National A.

4/ Same as National C.

3/ Same as National B.

5/ Same as National D.

Permissible explosives as of June 30, 1937 (continued)

NONGELATINOUS PERMISSIBLE EXPLOSIVES

Brand	Class designation basis, volume of poisonous gases ^{1/}	Weight of 1-1/4-inch cartridge	Small-diameter	Rate of detonation in 1-1/4-inch-diameter cartridge						Manufacturer
				Feet per second			Meters per second			
				Lower limit	Rate de-terminated	Upper limit	Lower limit	Rate de-terminated	Upper limit	
1	2	3	4	5						6
		Grams	Inches							
Big Red No. 7-C	A	160	7/8.	7,800	9,200	10,600	2,400	2,800	3,200	Equitable
Black Diamond No. 5-A	A	163	7/8	8,100	9,500	10,900	2,500	2,900	3,300	Illinois
Black Diamond No. 7	A	183	7/8	9,200	10,800	12,400	2,800	3,300	3,800	Do.
Black Diamond No. 7-A	B	155	7/8	9,800	11,500	13,200	3,000	3,500	4,000	Do.
Black Diamond No. 8	A	134	7/8	6,400	7,500	8,700	2,000	2,300	2,600	Do.
Black Diamond No. 9	B	144	7/8	7,200	8,500	9,800	2,200	2,600	3,000	Do.
Black Diamond No. 11	B	123	7/8	7,500	8,900	10,200	2,300	2,700	3,100	Do.
Black Diamond No. 11-A	B	111	7/8	8,600	10,200	11,700	2,600	3,100	3,600	Do.
Black Diamond No. 12	C	106	7/8	6,700	7,900	9,100	2,000	2,400	2,800	Do.
Black Diamond No. 12-A	C	100	7/8	7,200	8,500	9,800	2,200	2,600	3,000	Do.
Black Diamond No. 15	A	160	7/8	5,600	6,600	7,500	1,700	2,000	2,300	Do.
Black Diamond A	A	194	1	7,200	8,500	9,800	2,200	2,600	3,000	Do.
Black Diamond G	B	90	1-1/8	7,200	8,500	9,800	2,200	2,600	3,000	Do.
Black Diamond Special A ^{2/}	A	169	1-1/4	4,200	4,900	5,600	1,300	1,500	1,700	Do.
Black Diamond Special C ^{2/}	A	138	1-1/4	5,000	5,900	6,800	1,500	1,800	2,100	Do.
Burcoal 1, L. F.	A	110	1-1/8	5,000	5,900	6,800	1,500	1,800	2,100	Burton-General

^{1/} The classification on the basis of the volume of poisonous gases produced is given on page 2.

^{2/} Added to permissible list July, 1937.

Permissible explosives as of June 30, 1937 (continued)

NONGELATINOUS PERMISSIBLE EXPLOSIVES

Brand	Class designation basis, volume of poisonous gases	Weight of 1-1/4-inch cartridge	Small-diameter	Rate of detonation in 1-1/4-inch-diameter cartridge						Manufacturer	
				Feet per second			Meters per second				
				Lower limit	Rate terminated	Upper limit	Lower limit	Rate terminated	Upper limit		
				6							
1	2	3	4	5						Burton-General	
		Grams	Inches								Do.
Burcoal 2, L. F. 2/	B	99	1	5,000	5,900	6,800	1,500	1,800	2,100		
Burcoal 3, L. F. 3/	B	89	1-1/8	5,000	5,900	6,800	1,500	1,800	2,100		
Burcoal 4, L. F. 4/	A	128	1	5,000	5,900	6,800	1,500	1,800	2,100		
Burcoal 5, L. F. 5/	B	107	1	5,000	5,900	6,800	1,500	1,800	2,100		
Burton 1, L. F. 2/	B	111	1	8,900	10,500	12,100	2,700	3,200	3,700		
Burton 2, L. F.	A	143	7/8	8,900	10,500	12,100	2,700	3,200	3,700		
Burton 5, L. F.	A	143	7/8	6,100	7,200	8,300	1,900	2,200	2,500		
Burton 7, L. F.	A	134	7/8	6,400	7,500	8,700	2,000	2,300	2,600		
Burton 8, L. F.	A	160	7/8	5,000	5,900	6,800	1,500	1,800	2,100		
Burton 10, L. F.	B	124	1	7,500	8,900	10,200	2,300	2,700	3,100		
Burton 10-A	A	159	1	7,000	8,200	9,400	2,100	2,500	2,900		
Burton 11, L. F.	B	109	1	6,100	7,200	8,300	1,900	2,200	2,500		
Burton 12, L. F.	B	102	7/8	8,100	9,500	10,900	2,500	2,900	3,300		
Burton A, L. F.	B	201	1	6,100	7,200	8,300	1,900	2,200	2,500		
Burton G, L. F.	B	88	1	7,800	9,200	10,600	2,400	2,800	3,200		
Coalite A, L. F.	A	166	7/8	9,800	11,500	13,200	3,000	3,500	4,000		Atlas-Giant
Coalite B, L. F.	B	138	7/8	9,800	11,500	13,200	3,000	3,500	4,000		Do.

1/ The classification on the basis of the volume of poisonous gases produced is given on page 2.

2/ Transferred from inactive list.

4/ Same as Genite E.

3/ Transferred from inactive list.

NONGELATINOUS PERMISSIBLE EXPLOSIVES

Brand	Class designation basis, volume of poisonous gases ^{1/}	Weight of 1-1/4-inch cartridge	Small-est permissible diameter	Rate of detonation in 1-1/4-inch-diameter cartridge						Manufacturer
				Feet per second			Meters per second			
				Lower limit	Rate de-terminated limit	Upper limit	Lower limit	Rate de-terminated limit	Upper limit	
				5						
1	2	3	4	6						
		Grams	Inches							Atlas-Giant
Coalite E, L. F.	A	168	7/8	5,300	6,200	7,200	1,600	1,900	2,200	Do.
Coalite G, L. F.	A	152	7/8	6,700	7,900	9,100	2,000	2,400	2,800	Do.
Coalite K, L. F.	B	101	7/8	8,100	9,500	10,900	2,500	2,900	3,300	Do.
Coalite L, L. F.	A	111	7/8	7,500	8,900	10,200	2,300	2,700	3,100	Do.
Coalite LS, L. F.	B	103	7/8	5,300	6,200	7,200	1,600	1,900	2,200	Do.
Coalite M, L. F.	B	124	7/8	8,100	9,500	10,900	2,500	2,900	3,300	Do.
Coalite O, L. F.	B	92	7/8	6,400	7,500	8,700	2,000	2,300	2,600	Do.
Coalite S, L. F.	B	138	1	6,700	7,900	9,100	2,000	2,400	2,800	Do.
Coalite T, L. F.	A	139	7/8	5,000	5,900	6,800	1,500	1,800	2,100	Do.
Collier C, L. F.	A	141	7/8	10,000	11,800	13,600	3,100	3,600	4,100	Hercules
Columbia B	A	177	1	6,700	7,900	9,100	2,000	2,400	2,800	Columbia
Columbia C	A	150	1	7,000	8,200	9,400	2,100	2,500	2,900	Do.
Columbia D	A	142	7/8	6,400	7,500	8,700	2,000	2,300	2,600	Do.
Columbia E	A	129	1	6,400	7,500	8,700	2,000	2,300	2,600	Do.
Columbia F	A	116	1	7,500	8,900	10,200	2,300	2,700	3,100	Do.

^{1/} The classification on the basis of the volume of poisonous gases produced is given on page 2.

NONGELATINOUS PERMISSIBLE EXPLOSIVES

Brand	Class designation basis, volume of poisonous gases	Weight of 1- x 8-inch cartridge	Small-est permissible diameter	Rate of detonation in 1-1/4-inch-diameter cartridge								Manufacturer	
				Feet per second				Meters per second					
				Lower limit	Rate de-terminated	Upper limit	Lower limit	Rate de-terminated	Upper limit				
1	2	3	4	5				6					
		Grams	Inches										
Columbia G	A	114	7/8	4,700	5,600	6,400	1,400	1,700	2,000	Columbia			
Columbia 1	B	86	1	6,100	7,200	8,300	1,900	2,200	2,500	Do.			
Columbia 2	C	106	1	7,500	8,900	10,200	2,300	2,700	3,100	Do.			
Columbia 3	A	134	1	9,500	11,200	12,800	2,900	3,400	3,900	Do.			
Columbia 4	A	136	7/8	8,400	9,800	11,300	2,600	3,000	3,400	Do.			
Duobel No. 3, L. F.	B	110	7/8	8,600	10,200	11,700	2,600	3,100	3,600	du Pont			
Duobel No. 4, L. F.	B	111	7/8	6,700	7,900	9,100	2,000	2,400	2,800	Do.			
Duobel A	A	168	7/8	8,900	10,500	12,100	2,700	3,200	3,700	Do.			
Duobel B	A	150	7/8	8,400	9,800	11,300	2,600	3,000	3,400	Do.			
Duobel C	A	143	7/8	8,400	9,800	11,300	2,600	3,000	3,400	Do.			
Duobel D	A	124	7/8	8,600	10,200	11,700	2,600	3,100	3,600	Do.			
Duobel E	A	112	7/8	9,200	10,800	12,400	2,800	3,300	3,800	Do.			
Duobel F	B	99	7/8	7,800	9,200	10,600	2,400	2,800	3,200	Do.			
Duobel G	B	91	7/8	7,200	8,500	9,800	2,200	2,600	3,000	Do.			
Genite 22/	A	143	7/8	8,900	10,500	12,100	2,700	3,200	3,700	Burton-General			
Genite 42/	A	128	1	5,000	5,900	6,800	1,500	1,800	2,100	Do.			

1/ The classification on the basis of the volume of poisonous gases produced is given on page 2.

2/ Same as Burton 2, L. F.

3/ Same as Burcoal 4, L. F.

NONGELATINOUS PERMISSIBLE EXPLOSIVES

Brand	Class designation basis, volume of poisonous gases	Weight of 1-1/4-inch cartridge	Small-est permissible diameter	Rate of detonation in 1-1/4-inch-diameter cartridge						Manufacturer
				Feet per second			Meters per second			
				Lower limit	Rate terminated	Upper limit	Lower limit	Rate terminated	Upper limit	
				5			6			
Genite 5	A	143	Inches 7/8	6,100	7,200	8,300	1,900	2,200	2,500	Burton-General
Genite 7	A	134	7/8	6,400	7,500	8,700	2,000	2,300	2,600	Do.
Genite 10-A ²	A	159	1	7,000	8,200	9,400	2,100	2,500	2,900	Do.
Genite A	A	163	7/8	9,500	11,200	12,800	2,900	3,400	3,900	Do.
Genite B	B	141	1	9,500	11,200	12,800	2,900	3,400	3,900	Do.
Genite C-2	B	102	7/8	8,100	9,500	10,900	2,500	2,900	3,300	Do.
Genite D	B	109	1	6,100	7,200	8,300	1,900	2,200	2,500	Do.
Genite E	B	107	1	5,000	5,900	6,800	1,500	1,800	2,100	Do.
Genite F	E	143	1	7,000	8,200	9,400	2,100	2,500	2,900	Do.
Genite G	B	88	1	7,800	9,200	10,600	2,400	2,800	3,200	Do.
Genite H	A	160	7/8	5,000	5,900	6,800	1,500	1,800	2,100	Do.
Genite M	B	124	1	7,800	9,200	10,600	2,400	2,800	3,200	Do.
Grasselli 5, L. F.	A	156	7/8	6,400	7,500	8,700	2,000	2,300	2,600	du Pont
Hercoall C-1	A	117	1	7,800	9,200	10,600	2,400	2,800	3,200	Hercules
Hercoall D	B	98	1	7,500	8,900	10,200	2,300	2,700	3,100	Do.
Hercoall F-1	B	88	7/8	7,200	8,500	9,800	2,200	2,600	3,000	Do.

1/ The classification on the basis of the volume of poisonous gases produced is given on page 2.

2/ Same as Burton 10-A.

NONGELATINOUS PERMISSIBLE EXPLOSIVES

Brand	Class designation basis, volume of poisonous gases	Weight of 1-1/4-inch cartridge	Small-est permissible diameter	Rate of detonation in 1-1/4-inch-diameter cartridge						Manufacturer
				Feet per second			Meters per second			
				Lower limit	Rate terminated	Upper limit	Lower limit	Rate terminated	Upper limit	
1	2	3	4	5						6
Hercules Coal Powder 2	A	Grams 168	Inches 1-1/8	7,800	9,200	10,600	2,400	2,800	3,200	Hercules
King No. 5-A ^{2/}	A	163	7/8	8,100	9,500	10,900	2,500	2,900	3,300	King
King No. 7 ^{3/}	A	183	7/8	9,200	10,800	12,400	2,800	3,300	3,800	Do.
King No. 7-A ^{4/}	B	155	7/8	9,800	11,500	13,200	3,000	3,500	4,000	Do.
King No. 8 ^{5/}	A	134	7/8	6,400	7,500	8,700	2,000	2,300	2,600	Do.
King No. 9 ^{6/}	B	144	7/8	7,200	8,500	9,800	2,200	2,600	3,000	Do.
King No. 11 ^{7/}	B	123	7/8	7,500	8,900	10,200	2,300	2,700	3,100	Do.
King No. 11-A ^{8/}	B	111	7/8	8,600	10,200	11,700	2,600	3,100	3,600	Do.
King No. 12 ^{9/}	C	106	7/8	6,700	7,900	9,100	2,000	2,400	2,800	Do.
King No. 12-A ^{10/}	C	100	7/8	7,200	8,500	9,800	2,200	2,600	3,000	Do.
King No. 15 ^{11/}	A	160	7/8	5,600	6,600	7,500	1,700	2,000	2,300	Do.
King A ^{12/}	A	194	1	7,500	8,900	10,200	2,300	2,700	3,100	Do.
King G ^{13/}	B	90	1-1/8	7,200	8,500	9,800	2,200	2,600	3,000	Do.
King Special A ^{14/}	A	169	1-1/4	4,200	4,900	5,600	1,300	1,500	1,700	Do.

1/ The classification on the basis of the volume of poisonous gases produced is given on page 2.

2/ Same as Black Diamond No. 5-A. 6/ Same as Black Diamond No. 9. 10/ Same as Black Diamond No. 12-A.

3/ Same as Black Diamond No. 7. 7/ Same as Black Diamond No. 11. 11/ Same as Black Diamond No. 15.

4/ Same as Black Diamond No. 7-A. 8/ Same as Black Diamond No. 11-A. 12/ Same as Black Diamond A.

5/ Same as Black Diamond No. 8. 9/ Same as Black Diamond No. 12. 13/ Same as Black Diamond G.

14/ Same as Black Diamond Special A.

Permissible explosives as of June 30, 1937 (continued)

NONGELATINOUS PERMISSIBLE EXPLOSIVES

Brand	Class designation basis, volume of poisonous gases	Weight of 1-1/4-inch cartridge	Small-diameter	Rate of detonation in 1-1/4-inch-diameter cartridge						Manufacturer
				Feet per second			Meters per second			
				Lower limit	Rate terminated	Upper limit	Lower limit	Rate terminated	Upper limit	
1	2	3	4	5						6
		<u>Grams</u>	<u>Inches</u>							
Liberty No. 1	C	106	1	7,500	8,900	10,200	2,300	2,700	3,100	Liberty
Liberty No. 2	A	134	1	9,500	11,200	12,800	2,900	3,400	3,900	Do.
Liberty No. 2-A	A	142	7/8	6,400	7,500	8,700	2,000	2,300	2,600	Do.
Liberty No. 3	A	150	1	7,000	8,200	9,400	2,100	2,500	2,900	Do.
Liberty No. 4	B	171	7/8	6,100	7,200	8,300	1,900	2,200	2,500	Do.
Liberty No. 5	A	186	7/8	8,400	9,800	11,300	2,600	3,000	3,400	Do.
Liberty No. 6	A	116	1	7,500	8,900	10,200	2,300	2,700	3,100	Do.
Liberty No. 7	A	177	1	6,700	7,900	9,100	2,000	2,400	2,800	Do.
Liberty No. 8	A	129	1	6,400	7,500	8,700	2,000	2,300	2,600	Do.
Liberty No. 9	A	114	7/8	4,700	5,600	6,400	1,400	1,700	2,000	Do.
Liberty No. 11	B	86	1	6,100	7,200	8,300	1,900	2,200	2,500	Do.
Lump Coal A	A	165	1-1/4	4,700	5,600	6,400	1,400	1,700	2,000	du Pont
Lump Coal C	A	137	1-1/4	5,000	5,900	6,800	1,500	1,800	2,100	Do.
Miners Friend No. 1	B	167	7/8	9,800	11,500	13,200	3,000	3,500	4,000	Atlas
Miners Friend No. 2	B	166	7/8	8,900	10,500	12,100	2,700	3,200	3,700	Do.
Monobel No. 2, L. F.	A	175	7/8	9,200	10,800	12,400	2,800	3,300	3,800	du Pont
Monobel No. 6, L. F.	A	176	7/8	11,700	13,800	15,800	3,600	4,200	4,800	Do.

1/ The classification on the basis of the volume of poisonous gases produced is given on page 2.

Permissible explosives as of June 30, 1937 (continued)

NONGELATINOUS PERMISSIBLE EXPLOSIVES

Brand	Class designation basis, volume of poisonous gases ^{1/}	Weight of 1-1/4-inch cartridge	Small-est permissible diameter	Rate of detonation in 1-1/4-inch-diameter cartridge						Manufacturer
				Feet per second			Meters per second			
				Lower limit	Rate de-terminated	Upper limit	Lower limit	Rate de-terminated	Upper limit	
1	2	3	4	5						6
		Grams	Inches							
Monobel No. 8, L. F.	A	169	7/8	7,000	8,200	9,400	2,100	2,500	2,900	du Pont
Monobel No. 9, L. F.	A	134	7/8	5,300	6,200	7,200	1,600	1,900	2,200	Do.
Monobel No. 9-A, L. F.	A	142	7/8	6,400	7,500	8,700	2,000	2,300	2,600	Do.
Monobel No. 10, L. F.	B	124	7/8	7,000	8,200	9,400	2,100	2,500	2,900	Do.
Monobel No. 12, L. F.	B	99	7/8	7,800	9,200	10,600	2,400	2,800	3,200	Do.
Monobel A	A	169	7/8	7,000	8,200	9,400	2,100	2,500	2,900	Do.
Monobel B	A	150	7/8	7,200	8,500	9,800	2,200	2,600	3,000	Do.
Monobel C ^{2/}	A	137	7/8	7,000	8,200	9,400	2,100	2,500	2,900	Do.
Monobel D	B	123	7/8	7,000	8,200	9,400	2,100	2,500	2,900	Do.
Monobel E	B	111	7/8	7,000	8,200	9,400	2,100	2,500	2,900	Do.
National A	A	160	1	7,200	8,500	9,800	2,200	2,600	3,000	National
National B	A	147	1	8,900	10,500	12,100	2,700	3,200	3,700	Do.
National C	B	100	1	7,500	8,900	10,200	2,300	2,700	3,100	Do.
National D	A	162	1	5,900	6,900	7,900	1,800	2,100	2,400	Do.
Peerless No. 1	B	148	7/8	9,500	11,200	12,800	2,900	3,400	3,900	Atlas
Peerless No. 8, L. F.	B	126	1-1/8	7,800	9,200	10,600	2,400	2,800	3,200	Do.

^{1/} The classification on the basis of the volume of poisonous gases produced is given on page 2.^{2/} New basic data as of April 12, 1937.

NONGELATINOUS PERMISSIBLE EXPLOSIVES

Brand	Class designation basis, volume of poisonous gases	Weight of 1-1/4-inch cartridge	Small-diameter	Rate of detonation in 1-1/4-inch-diameter cartridge						Manufacturer	
				Feet per second			Meters per second				
				Lower limit	Rate terminated	Upper limit	Lower limit	Rate terminated	Upper limit		
1	2	3	4	5						6	
		Grams	Inches								
Red HB, L. F.	A	155	7/8	10,300	12,100	14,000	3,100	3,700	4,300	Hercules	
Red HC, L. F.	A	168	7/8	5,000	5,900	6,800	1,500	1,800	2,100	Do.	
Red HD, L. F.	A	150	7/8	6,700	7,900	9,100	2,000	2,400	2,800	Do.	
Red HF, L. F.	A	130	1	6,400	7,500	8,700	2,000	2,300	2,600	Do.	
Red H, No. 4	B	164	7/8	7,500	8,900	10,200	2,300	2,700	3,100	Do.	
Red H No. 5	B	170	1	5,900	6,900	7,900	1,800	2,100	2,400	Do.	
Trojan Coal Powder M-2	A	178	1	8,900	10,500	12,100	2,700	3,200	3,700	Trojan	
Trojan Coal Powder M-3	A	184	1	10,300	12,100	14,000	3,100	3,700	4,300	Do.	
Trojan Coal Powder M-5	A	188	1	10,900	12,800	14,700	3,300	3,900	4,500	Do.	
Trojan Coal Powder M-6	A	190	1	10,000	11,800	13,600	3,100	3,600	4,100	Do.	
Trojan Coal Powder P-1	B	154	1-1/4	8,100	9,500	10,900	2,500	2,900	3,300	Do.	
Trojan Coal Powder P-2	B	174	1	10,900	12,800	14,700	3,300	3,900	4,500	Do.	
Trojan Coal Powder P-3	B	176	1	11,200	13,100	15,100	3,400	4,000	4,600	Do.	
Unilite No. 1	B	114	1-1/4	7,200	8,500	9,800	2,200	2,600	3,000	Atlas	
Unilite No. 2	B	103	1-1/4	7,000	8,200	9,400	2,100	2,500	2,900	Do.	
U. S. No. 5-A	A	163	7/8	8,100	9,500	10,900	2,500	2,900	3,300	United	

1/ The classification on the basis of the volume of poisonous gases produced is given on page 2.

Permissible explosives as of June 30, 1937 (continued)

NONGELATINOUS PERMISSIBLE EXPLOSIVES

Brand	Class designation basis, volume of poisonous gases ^{1/}	Weight of 1-1/4-inch cartridge	Small-diameter permissible diameter	Rate of detonation in 1-1/4-inch-diameter cartridge						Manufacturer	
				Feet per second			Meters per second				
				Lower limit	Rate determined	Upper limit	Lower limit	Rate determined	Upper limit		
1	2	3	4	5						6	
		<u>Grams</u>	<u>Inches</u>								
U. S. No. 7	A	183	7/8	9,200	10,800	12,400	2,800	3,300	3,800	United	
U. S. No. 7-A	B	155	7/8	9,800	11,500	13,200	3,000	3,500	4,000	Do.	
U. S. No. 8	A	134	7/8	6,400	7,500	8,700	2,000	2,300	2,600	Do.	
U. S. No. 9	B	144	7/8	7,200	8,500	9,800	2,200	2,500	3,000	Do.	
U. S. No. 11	B	123	7/8	7,500	8,900	10,200	2,300	2,700	3,100	Do.	
U. S. No. 11-A	B	111	7/8	8,600	10,200	11,700	2,600	3,100	3,600	Do.	
U. S. No. 12	C	106	7/8	6,700	7,900	9,100	2,000	2,400	2,800	Do.	
U. S. No. 12-A	C	100	7/8	7,200	8,500	9,800	2,200	2,600	3,000	Do.	
U. S. No. 15	A	160	7/8	5,600	6,600	7,500	1,700	2,000	2,300	Do.	
U. S. A	A	194	1	7,200	8,500	9,800	2,200	2,600	3,000	Do.	
U. S. G	B	90	1-1/8	7,200	8,500	9,800	2,200	2,600	3,000	Do.	

^{1/} The classification on the basis of the volume of poisonous gases produced is given on page 2.

Permissible explosives as of June 30, 1937 (continued)

GELATINOUS PERMISSIBLE EXPLOSIVES

Brand	Class designation basis, volume of poisonous gases	Weight of 1/4-inch cartridge	Small-est permissible diameter	Rate of detonation in 1-1/4-inch-diameter cartridge						Manufacturer
				Feet per second		Meters per second				
				Lower limit	Rate de-terminated	Upper limit	Rate de-terminated	Upper limit		
				Lower limit	Rate de-terminated	Upper limit	Rate de-terminated	Upper limit		
1	2	3	4	5						6
		Grams	Inches							
Austin Red-D-Gel	A	244	1	12,500	14,800	17,000	3,800	4,500	5,200	Austin
Black Diamond Nu-Gel No. 1	A	244	7/8	11,400	13,400	15,500	3,500	4,100	4,700	Illinois
Black Diamond Nu-Gel No. 2	A	232	7/8	9,200	10,800	12,400	2,300	3,300	3,800	Do.
Black Diamond Nu-Gel No. 4	A	193	7/8	11,400	13,400	15,500	3,500	4,100	4,700	Do.
Columbia-Gel 1	A	226	1	11,700	13,800	15,800	3,600	4,200	4,800	Columbia
Columbia-Gel 2	A	171	7/8	8,600	10,200	11,700	2,600	3,100	3,600	Do.
Gel-Coalite W, L. F.	A	139	7/8	10,600	12,500	14,300	3,200	3,800	4,400	Atlas-Giant
Gel-Coalite X, L. F.	A	239	7/8	8,100	9,500	10,900	2,500	2,900	3,300	Do.
Gel-Coalite Z, L. F.	A	226	7/8	14,200	16,700	19,200	4,300	5,100	5,900	Do.
Gelobel, L. F.	A	236	7/8	12,800	15,100	17,400	3,900	4,600	5,300	du Pont
Gelobel, No. 3, L. F.	A	247	1	6,700	7,900	9,100	2,000	2,400	2,800	Do.
Gelobel No. 4, L. F.	A	190	7/8	11,400	13,400	15,500	3,500	4,100	4,700	Do.
Gen-Gel No. 2	A	217	1	8,400	9,300	11,300	2,600	3,000	3,400	Burton-Gener
Gen-Gel No. 3	A	183	1	10,600	12,500	14,300	3,200	3,800	4,400	Do.

1/ The classification on the basis of the volume of poisonous gases produced is given on page 2.

2/ Rates of detonation of gelatinous permissible explosives may be low (approximately 2,000 m/s), high (approximately 5,000 m/s), or intermediate. However, in a borehole it is believed that all of these explosives will develop rates close to the high rate.

Permissible explosives as of June 30, 1937 (continued)

GELATINOUS PERMISSIBLE EXPLOSIVES

Brand	Class designation basis, volume of poisonous gases	Weight of 1-1/4-inch cartridge	Small-est permissible diameter	Rate of detonation in 1-1/4-inch-diameter cartridge						Manufacturer	
				Feet per second		Meters per second					
				Lower limit	Rate determined	Upper limit	Lower limit	Rate determined	Upper limit		
1	2	3	4	5						6	
		Grams	Inches								
Hercogel 2	A	186	7/8	10,000	11,800	13,600	3,100	3,600	4,100	Hercules	
Hercogel A	A	220	7/8	12,500	14,800	17,000	3,800	4,500	5,200	Do.	
King Nu-Gel No. 1 ^{3/4}	A	244	7/8	11,400	13,400	15,500	3,500	4,100	4,700	King	
King Nu-Gel No. 2 ^{4/5}	A	232	7/8	9,200	10,800	12,400	2,800	3,300	3,800	Do.	
King Nu-Gel No. 4 ^{5/6}	A	193	7/8	11,400	13,400	15,500	3,500	4,100	4,700	Do.	
Liberty-Gel, L. F.	A	226	1	11,700	13,800	15,800	3,600	4,200	4,800	Liberty	
Liberty-Gel B	A	171	7/8	8,600	10,200	11,700	2,600	3,100	3,600	Do.	
Permigel 1, L. F.	A	238	1	11,700	13,800	15,800	3,600	4,200	4,800	Burton-General	
Permigel 2, L. F.	A	225	7/8	7,800	9,200	10,600	2,400	2,800	3,200	Do.	
Permigel 3, L. F.	A	183	1	10,600	12,500	14,300	3,200	3,800	4,400	Do.	
U. S. Nu-Gel No. 1	A	244	7/8	11,400	13,400	15,500	3,500	4,100	4,700	United	
U. S. Nu-Gel No. 2	A	232	7/8	9,200	10,800	12,400	2,800	3,300	3,800	Do.	

1/ The classification on the basis of the volume of poisonous gases produced is given on page 2.

2/ See p. 15, footnote 2.

4/ Same as Black Diamond Nu-Gel No. 2.

3/ Same as Black Diamond Nu-Gel No. 1.

5/ Same as Black Diamond Nu-Gel No. 4.

6/ Transferred from inactive list.

It is provided further:

That, in accordance with Schedule 17C, under which the previously named explosives have been tested, explosives enumerated on the permissible lists of the United States Bureau of Mines are permissible in use only when they satisfy the following requirements:

1. That the explosive is in all respects similar to the sample submitted by the manufacturer for test.

2. That electric detonators (not fuse and detonators) are used of not less efficiency than no. 6, the detonating charge of which shall consist by weight of 80 parts of mercury fulminate and 20 parts of potassium chlorate (or their equivalents) - and that the required electric firing must be done by means of a permissible-type blasting unit.

3. That the explosive, if frozen, shall be thawed thoroughly in a safe and suitable manner before use.

4. That the quantity used for a shot does not exceed 680 grams (1-1/2 pounds), and that it is properly confined with clay or other incombustible stemming.

5. That the diameter of the cartridge used must be not less than that designated in the column "Smallest permissible diameter."

6. That the shot is not fired in the presence of a dangerous percentage of firedamp.

7. That the shot is not a dependent shot, is not bored into the solid, and does not have a burden so heavy that the shot obviously is liable to blow out.

8. That the explosive be stored under proper conditions so that it does not undergo change in character.

The complete name and address of each manufacturer indicated in the table is as follows:

Apache:	Apache Powder Co., Benson, Ariz.
Atlas:	Atlas Powder Co., Wilmington, Del.
Austin:	Austin Powder Co., Cleveland, Ohio.
Burton:	Burton Explosives Division of American Cyanamid & Chemical Corporation, Cleveland, Ohio.
Columbia:	Columbia Powder Co., Tacoma, Wash.
du Pont:	E. I. du Pont de Nemours & Co., Wilmington, Del.
Equitable:	Equitable Powder Mfg. Co. and Egyptian Powder Co., East Alton, Ill.
General:	General Explosives Division of American Cyanamid & Chemical Corporation, Latrobe, Pa.
Giant:	Giant Powder Co. (Con.), Giant, Calif.
Hercules:	Hercules Powder Co., Wilmington, Del.
Illinois:	Illinois Powder Mfg. Co., St. Louis, Mo.
Independent:	Independent Explosives Co. of Pa., Pittston, Pa.
King:	King Powder Co., Inc., Cincinnati, Ohio.
Liberty:	Liberty Powder Co., Pittsburgh, Pa.
National:	National Powder Co., Duke Center, Pa.
Trojan:	Trojan Powder Co., Allentown, Pa.
United:	United States Powder Co., Herre Haute, Ind.

PERMISSIBLE BLASTING DEVICES TESTED UNDER SCHEDULE 20

An active list of permissible blasting devices was published in December 1936 as part of United States Bureau of Mines Report of Investigations 3324, issued under the title "Active List of Permissible Explosives and Blasting Devices Approved Prior to July 31, 1936."

The permissible blasting devices Cardox models B-20, B-37, B-44, C-80, D-100, 2-80, 2-100 and 3-200 utilize as a blasting agent liquid carbon dioxide which is confined in a stout steel cylinder with a heater element which by its combustion supplies enough energy to gasify the carbon dioxide. One end of the steel cylinder is closed hermetically by a replaceable steel disk which is sheared by the pressure of the carbon dioxide when sufficiently gasified, and the latter is thus enabled to escape from the cylinder. When blasting with Cardox the heater is fired by means of a permissible blasting unit. The permissibility of the Cardox devices was established by gallery tests similar to those used in testing explosives. In these tests the shell was discharged into gas and air, or gas, coal dust, and air mixtures without igniting them. The conditions of charging the shell, such as the weight of the carbon dioxide charge and of the ingredients of the heater element, and the thickness of the disk, were varied according to the conditions under which these devices would be likely to be used in blasting coal. Besides the gallery tests, tests have been made to determine the propulsive strength of the devices and the quantities of poisonous gases given off when a shell is discharged.

The complete active list of permissible blasting devices approved prior to June 30, 1937 is shown in the following table, which includes the permissible conditions of charging each shell for a given thickness of disk as to maximum weight of heater ingredient.

Permissible Cardox models as of June 30, 1937 Manufactured by Safety Mining Co., Chicago, Ill.

Authorized combinations									
Number of	Approval	Extension	Dimension of shell		In.	In.	In.	Grams	Thousands of pounds
			Length	Diameter					
CARDON MODEL B-20									
7	1, 2, and 3	60.7	23-7/8	4.4	1-3/4	Usual charge of carbon dioxide 255 grams	4/	30	14
								30	14
								35	12
								35	12
								45	10
								45	10
CARDON MODEL B-37									
6	2, 3, and 4	103.2	40-5/8	4.4	1-3/4	Usual charge of carbon dioxide 400	4/	45	16
								45	16
								60	14
								60	14
								80	12
								80	12
								80	12
								110	10
								110	10
								110	8
								110	8
CARDON MODEL B-44									
8	2, 3, and 4	118.4	46-5/8	4.4	1-3/4	Usual charge of carbon dioxide 570 grams	4/	42	16
								42	16
								80	14
								80	14
								100	12
								100	12
								110	10
								110	10
								110	8
								110	8
CARDON MODEL C-80									
4		98.6	38-13/16	6.4	2-1/2	Usual charge of carbon dioxide 1,075 grams	5/	6/140	11
								6/140	9
								6/140	7

A u t h o r i z e d c o m b i n a t i o n s 1/

N u m b e r o f	Approval	Extension	Dimension of shell		Length	Diameter		Shear opening in discharge cap, diameter	Heater, maximum weight of ingredient (potassium-perchlorate-charcoal mixture 2/4/	Disk (sheet metal)	Thousands of pou	
			Cm.	In.		Cm.	In.					Minimum gage pressure 3/
5	3, 4, and 5	118.1	46-1 1/2	6.4	2-1 1/2	CARDON MODEL D-100					16 11 21 17 25 21	
						Usual charge of carbon dioxide 1,425 grams 4/						
						1-1/4	180					7/12
						1-7/16	180					7/12
						1-1/4	200					7/10
10		118.4	46-5/8	5.1	2	CARDON MODEL 2-80					8 10 13 17	
						Usual charge of carbon dioxide 1,075 grams 4/						
						1-1/16	80					16
						1-1/16	110					14
						1-1/16	110					12
11		148.9	58-5/8	5.1	2	CARDON MODEL 2-100					8 10 13 17	
						Usual charge of carbon dioxide 1,425 grams 4/						
						1-1/16	110					16
						1-1/16	110					14
						1-1/16	110					12
9		134.0	52-3/4	7.6	3	CARDON MODEL 3-200					10 13 17	
						Usual charge of carbon dioxide 2,800 grams 4/						
						1-3/4	220					7/12
						1-3/4	220					7/10
						1-3/4	220					7/8

1/ The maximum weight of heater and minimum disk gage are combinations for the usual charge of carbon dioxide as stated for each model. With the exception of Cardox Model C-80, the combinations studied and tested at the gallery are for charges of carbon dioxide that will not under normal conditions shear the disk; therefore a charge of carbon dioxide less than that usually used does not affect the permissible use of the device. For Cardox Model C-80 the minimum charge of carbon dioxide that can be used is 850 grams. Except for Cardox Model C-80 the ingredients of the heater are potassium perchlorate and charcoal. The heater approved for Cardox Model C-80 contains, in addition to these ingredients, aluminum. The minimum pressures allow a reasonable tolerance from those actually determined by tests. The tests were made with charges of carbon dioxide that would not under normal conditions shear the disk, thus affording safety against leakage of carbon dioxide. Minimum carbon dioxide charge that can be used is 850 grams. This heater contains potassium perchlorate, charcoal, and aluminum. 1/ Strip steel disks instead of sheet metal.

GENERAL REQUIREMENTS

It is provided further:

That in accordance with Schedule 20 under which the above-named permissible blasting devices have been tested, these devices are permissible only when satisfying the following requirements:

1. That the device is in all respects similar to the sample submitted for tests.
2. That the permissible conditions of charging as to thickness of disk, weight of heater ingredient, and weight of carbon dioxide charge are met, and that the device is properly confined with clay or other noncombustible stemming.
3. That the lead wires shall be connected to either type of shell top (end or side connection) only after the shell has been placed in the borehole and further, that the terminal plug when the end connection shell top is used, shall not be inserted into the shell until these connections are being made.
4. That the device shall not be fired in the presence of a dangerous percentage of fire damp.
5. That the device shall not be charged with the heater ingredient and carbon dioxide in the mine.

Except for Cardox Model C-80, all models use a heater mixture containing potassium perchlorate and charcoal. Model C-80 uses a heater containing aluminum in addition to potassium perchlorate and charcoal.

JOHN W. FINCH,
Director.

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PROPERTIES OF CALIFORNIA CRUDE OILS

V - ADDITIONAL ANALYSES

UNIVERSITY OF ILLINOIS



BY

E. C. LANE AND E. L. GARTON

REPORTS OF INVESTIGATIONS

UNITED STATES DEPARTMENT OF THE INTERIOR - BUREAU OF MINES

PROPERTIES OF CALIFORNIA CRUDE OILS

V - Additional Analyses^{1/}

By E. C. Lane^{2/} and E. L. Garton^{3/}

INTRODUCTION

This paper on the physical and chemical properties of crude oils from California fields is one of a series of reports on the oil-producing districts of the United States. The paper includes a short discussion of results, tabular data, and individual analyses of 16 California crude oils. Analyses of other oils from California have been published in Reports of Investigations 2595^{4/}, 2608^{5/}, 2846^{6/}, and 3074^{7/}.

The samples were obtained by members of the San Francisco office of the Bureau of Mines. The collecting of the samples was facilitated by the courtesy of representatives of oil companies in California.

The methods used in testing and analyzing the samples are described in detail in Bureau of Mines Bulletin 207^{8/}.

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- 1/ The Bureau of Mines will welcome reprinting of this paper provided the following footnote acknowledgment is used: "Reprinted from Bureau of Mines Report of Investigations 3362."
 - 2/ Assistant petroleum chemist, Bureau of Mines, Bartlesville, Okla.
 - 3/ Assistant physical chemist, Bureau of Mines, Bartlesville, Okla.
 - 4/ Kraemer, A. J., and Smith, H. M., Properties of Typical Crude Oils from the Producing Fields of California: Rept. of Investigations 2595, Bureau of Mines, 1924, 55 pp.
 - 5/ Kraemer, A. J., and Smith, H. M., Properties of California Crude Oils. II - Additional Analyses: Rept. of Investigations 2608, Bureau of Mines, 1924, 55 pp.
 - 6/ Kraemer, A. J., Properties of California Crude Oils. III - Additional Analyses: Rept. of Investigations 2846, Bureau of Mines, 1927, 27 pp.
 - 7/ Kraemer, A. J., Properties of California Crude Oils. IV - Additional Analyses: Rept. of Investigations 3074, Bureau of Mines, 1931, 12 pp.
 - 8/ Dean, E. W., and others, The Analytical Distillation of Petroleum and Its Products: Bull. 207, Bureau of Mines, 1922, 82 pp.

DISCUSSION OF RESULTS

Table 1 summarizes the results of these analyses and presents, for comparison, some of the figures indicating the physical and chemical properties of the crude oils. Data used in determining the "base" of the oils, according to the method of Lane and Garton^{9/}, are shown in table 2. The following statement is quoted from their report.

Method of Determining the "Base" of a Crude Oil

1. Note the A.P.I. gravity of "key fraction no. 1", the fraction distilling between 250° and 275°C. (482° and 527°F.) at atmospheric pressure.

2. Note the A.P.I. gravity of "key fraction no. 2", the fraction distilling between 275° and 300° C. (527° and 572° F.) at 40 mm absolute pressure.

For purposes of classification, the gravity of key fraction no. 1 is taken as representing the lighter (lower-boiling) fractions of the oil, whereas the gravity of key fraction no. 2 represents the heavier (higher-boiling) distillates.

If the gravity of key fraction no. 1 is 40.0° A.P.I. or lighter, the lower-boiling fractions of the oil are paraffinic in character; if it is 33.0° A.P.I. or heavier, they are naphthenic; and if it is between 33.0° and 40.0° (33.1° to 39.9° A.P.I., inclusive), they are intermediate.

Similarly, if the gravity of key fraction no. 2 is 30.0° A.P.I. or lighter, the higher-boiling fractions of the oil are paraffinic in character. They are naphthenic if key fraction no. 2 is 20.0° A.P.I. or heavier and intermediate if the key fraction is between 20 and 30 (20.1° to 29.9° A.P.I., inclusive).

RELATION OF "BASE" TO FORMATION DEPTH

Two general types of crude oil are represented in this series of analyses. As shown in table 2, six samples (three from Kern County and three from Los Angeles County) are "naphthenic" throughout. Five of the six analyses show no indication that wax is present; the sixth, analysis 37167, apparently contains a small amount of wax. It is interesting to note that at least five of these six samples come from relatively shallow production. The sixth one is a composite pipe-line sample, so the depth of production cannot be stated definitely. Nine of the remaining ten analyses are considered as falling within the "intermediate" classification. The relationship between the character of the oil and the depth of production is again evident, as all of these nine samples come from comparatively deep structures. Sample 31010, from Kettleman North Dome,

^{9/} Lane, E. C., and Garton, E. L., "Base" of a Crude Oil: Rept. of Investigations 3279, Bureau of Mines, September 1935, 12 pp.

is of interest, as more than 90 percent of it distills in the "gasoline and naphtha" fraction. Comparison of the gravities of the distillation fractions shows that this crude oil is almost identical with the lighter fractions of the other samples from Kettleman North Dome, so it probably should be given the same classification as to "base".

Another interesting sample is 37169 from the Fruitvale field, which contains enough waxy hydrocarbons so that although the lighter distillates are "naphthenic" the heavier ones are "intermediate" in character.

TABLE 1. - Data indicating some physical and chemical properties

Sample	Field	A.P.I.	Sulphur percent,	Gasoline fraction Percent A.P.I.	Viscosity at 100°F. of vacuum fractions	Carbon residue of resi- dum, %		
							225°-250° C.	250°-275° C.
Kern County:								
35285	Edison	21.5	0.57	7.7	75	145	290	14.2
35286	do.	15.9	.68	-	92	190	Over 400	13.5
37196	Midway (light)	25.6	.65	21.2	77	160	do.	12.5
35287	Mountain View	30.0	.44	22.9	63	100	199	14.6
31044	North Belridge	42.6	(1)	44.2	58	93	180	8.3
37001	Greeley	36.8	.28	33.2	58	91	170	9.5
37002	Ten Section	37.8	.37	37.6	62	120	230	12.4
Kings County:								
31040	Kettleman North Dome	61.8	(1)	92.5	-	-	-	-
31041	do.	37.8	.33	39.0	60	95	210	11.2
31042	do.	42.3	.22	50.9	62	105	210	9.3
37158	do.	54.0	.50	31.9	65	97	200	12.4
Los Angeles County:								
31043	Del Rey	24.5	2.70	20.9	58	37	180	14.4
37169	Fruitvale	24.2	.60	10.1	66	115	240	11.8
37170	do.	16.2	.96	0.3	96	240	Over 400	11.9
37166	Wilmington	17.1	2.47	5.4	72	135	340	15.5
37167	do.	25.6	1.24	20.4	77	155	400	9.0
1/ Less than 0.10								

TABLE 2. -- Data indicative of the "base" of the crude petroleum

Sample	Field	Zone depth, feet	"Key fraction no. 1"		"Key fraction no. 2"		Indication of wax	Base of crude oil
			°A.P.I.		°A.P.I.	Cloud point		
35285	Kern County:	2,904-3,100	31.9		19.7	Below 5	Absent	Naphthene.
35286	Edison	1,577-1,668	31.1		17.3	do.	do.	Do.
37196	do.	Pipe Line	31.1		19.4	do.	do.	Do.
35287	Midway	5,505-5,560	35.2		23.0	95	Present	Intermediate.
31044	Mountain View	4,970-5,457	36.8		26.3	90	do.	Do.
37001	North Belridge	7,740-7,807	38.0		24.9	90	do.	Do.
37002	Greeley	7,763-8,383	35.4		22.8	90	do.	Do.
31040	Ten Section							
31041	Kings County:							
31042	Kettleman North-Dome	7,276						
37168	do.	7,065-8,323	36.8		24.5	90	Present	Intermediate.
	do.	6,894-7,904	37.0		24.9	90	do.	Do.
	do.	8,130-8,557	36.2		24.3	90	do.	Do.
31043	Los Angeles County:							
37169	Bel Rey	3,516-4,569	36.2		23.5	75	do.	Do.
37170	Fruitvale	3,754-3,924	31.5		23.5	90	do.	Naphthene intermedi
37166	do.	3,403-3,472	29.3		16.7	Below 5	Absent	Naphthene.
37167	Wilmington	3,190-3,618	31.5		19.7	do.	do.	Do.
	do.	3,109-3,905	32.8		19.7	55	Present	Do.

Sample 35285

2904-3100 feet

Edison Field
Lower Duff.California
Kern County
30S-29E M.D. B. & M.

GENERAL CHARACTERISTICS

Specific gravity, 0.925

A.P.I. gravity, 21.5°

Percent sulphur, 0.57

Color, brownish black

Saybolt Universal viscosity at 100°F., 200 seconds

Saybolt Universal viscosity at 130°F., 115 seconds

DISTILLATION, BUREAU OF MINES HEMPEL METHOD

Dry distillation

Barometer, 746 mm.

First drop: 85°C. (185°F.)

Temperature, °C.	Per- cent cut	Sum, per- cent	Sp. gr. of cut	°A.P.I. of cut	Viscosity at 100°F.	Cloud test, °F.	Temperature, °F.
Up to 50							Up to 122
50 - 75							122 - 167
75 - 100							167 - 212
100 - 125	0.7	0.7					212 - 257
125 - 150	1.6	2.3	0.756	55.7			257 - 302
150 - 175	2.6	4.9	.775	51.1			302 - 347
175 - 200	2.8	7.7	.800	45.4			347 - 392
200 - 225	3.8	11.5	.825	40.0			392 - 437
225 - 250	5.7	17.2	.847	35.6			437 - 482
250 - 275	8.2	25.4	.866	31.9			482 - 527

Vacuum distillation at 40 mm

Up to 200	4.5	4.5	0.889	27.7	43	Below 5	Up to 392
200 - 225	6.7	11.2	.902	25.4	53	do.	392 - 437
225 - 250	5.9	17.1	.915	23.1	75	do.	437 - 482
250 - 275	5.6	22.7	.926	21.3	145	do.	482 - 527
275 - 300	8.2	30.9	.936	19.7	290	do.	527 - 572

Carbon residue of residuum, 14.2 percent.

Carbon residue of crude, 5.8 percent.

APPROXIMATE SUMMARY

	Percent	Sp. gr.	°A.P.I.	Viscosity
Light gasoline				
Total gasoline and naphtha	7.7	0.778	50.4	
Kerosene distillate	3.8	.825	40.0	
Gas oil	20.1	.868	31.5	
Nonviscous lubricating distillate	10.0	.898-.919	26.1-22.5	50-100
Medium lubricating distillate	6.3	.919-.930	22.5-20.7	100-200
Viscous lubricating distillate	8.4	.930-.942	20.7-18.7	Above 200
Residuum	41.0	.996	10.6	
Distillation loss	2.7	-	-	

6000

Sample 35286

1577-1668 feet

Edison Field
Kern RiverCalifornia
Kern County
30S-29E M. D. B & M

GENERAL CHARACTERISTICS

Specific gravity, 0.960
 Percent sulphur, 0.68
 Saybolt Universal viscosity at 100°F., 1930 seconds
 Saybolt Universal viscosity at 130°F., 670 seconds

A.P.I. gravity, 15.9°
 Color, brownish black

DISTILLATION, BUREAU OF MINES HEMPEL METHOD

Dry distillation		Barometer, 752 mm.		First drop: 209°C. (408°F.)			
Temperature, °C.	Per- cent cut	Sum per- cent	Sp. gr. of cut	°A.P.I. of cut	Viscosity at 100°F.	Cloud test, °F.	Temperature °F.
Up to 50							Up to 122
50 - 75							122 - 167
75 - 100							167 - 212
100 - 125							212 - 257
125 - 150							257 - 302
150 - 175							302 - 347
175 - 200							347 - 392
200 - 225	0.9	0.9	0.829	39.2			392 - 437
225 - 250	2.9	3.8	.849	35.2			437 - 482
250 - 275	6.1	9.9	.870	31.1			482 - 527

Vacuum distillation at 40 mm

Up to 200	4.1	4.1	0.902	25.4	49	Below 5	Up to 392
200 - 225	6.6	10.7	.911	23.8	61	do.	392 - 437
225 - 250	7.3	18.0	.927	21.1	92	do.	437 - 482
250 - 275	7.7	25.7	.939	19.2	190	do.	482 - 527
275 - 300	9.7	35.4	.951	17.3	Over 400	do.	527 - 572

Carbon residue of residuum, 13.5 percent. Carbon residue of crude, 7.1 percent.

APPROXIMATE SUMMARY

	Percent	Sp. gr.	°A.P.I.	Viscosity
Light gasoline				
Total gasoline and naphtha				
Kerosene distillate				
Gas oil	12.4	0.868	31.5	
Nonviscous lubricating distillate	12.5	.903-.928	25.2-21.0	50-100
Medium lubricating distillate	7.3	.928-.940	21.0-19.0	100-200
Viscous lubricating distillate	13.1	.940-.958	19.0-16.2	Above 200
Residuum	52.5	1.001	-	
Distillation loss	2.2	-	-	

Sample 37196

Pipe Line Run

Midway Field (light)

California
Kern County

GENERAL CHARACTERISTICS

Specific gravity, 0.901

A.P.I. gravity, 25.6°

Percent sulphur, 0.65

Color, brownish black

Saybolt Universal viscosity at 77°F., 110 seconds

Saybolt Universal viscosity at 100°F., 75 seconds

DISTILLATION, BUREAU OF MINES HEMPEL METHOD

Dry distillation

Barometer, 745 mm.

First drop: 74°C. (165°F.)

Temperature, °C.	Per- cent cut	Sum per- cent	Sp. gr. of cut	°A.P.I. of cut	Viscosity at 100°F.	Cloud test, °F.	Temperature, °F.
Up to 50							Up to 122
50 - 75							122 - 167
75 - 100	1.8	1.8	0.712	67.2			167 - 212
100 - 125	4.9	6.7	.753	56.4			212 - 257
125 - 150	5.5	12.2	.776	50.9			257 - 302
150 - 175	4.5	16.7	.798	45.8			302 - 347
175 - 200	4.5	21.2	.819	41.3			347 - 392
200 - 225	5.2	26.4	.837	37.6			392 - 437
225 - 250	6.3	32.7	.853	34.4			437 - 482
250 - 275	7.5	40.2	.870	31.1			482 - 527

Vacuum distillation at 40 mm

Up to 200	4.6	4.6	0.888	27.9	43	Below 5	Up to 392
200 - 225	5.3	9.9	.900	25.7	53	do.	392 - 437
225 - 250	4.8	14.7	.914	23.3	77	do.	437 - 482
250 - 275	4.9	19.6	.928	21.0	160	do.	482 - 527
275 - 300	6.4	26.0	.938	19.4	Over 400	do.	527 - 572

Carbon residue of residuum, 12.5 percent. Carbon residue of crude, 4.1 percent.

APPROXIMATE SUMMARY

	Percent	Sp. gr.	°A.P.I.	Viscosity
Light gasoline	1.8	0.712	67.2	
Total gasoline and naphtha	21.2	.779	50.1	
Kerosene distillate	-	-	-	
Gas oil	24.8	.863	32.5	
Nonviscous lubricating distillate	7.9	.896-.918	26.4-22.6	50-100
Medium lubricating distillate	4.4	.918-.930	22.6-20.7	100-200
Viscous lubricating distillate	7.9	.930-.943	20.7-18.6	Above 200
Residuum	32.9	.987	11.9	
Distillation loss	.9	-	-	

Sample 35287

5505-5560 feet

Mountain View (California) Field

California

Kern County

30S-28E. M.D. B. & M

GENERAL CHARACTERISTICS

Specific gravity, 0.876

A.P.I. gravity, 30.0°

Percent sulphur, 0.44

Color, brownish black

Saybolt Universal viscosity at 77°F., 84 seconds

Saybolt Universal viscosity at 100°F., 56 seconds

DISTILLATION, BUREAU OF MINES HEMPEL METHOD

Dry distillation

Barometer, 746 mm.

First drop: 75°C. (167°F.)

Temperature, °C.	Per- cent cut	Sum, per- cent	Sp. gr. of cut	°A.P.I. of cut	Viscosity at 100°F.	Cloud test, °F.	Temperatur °F.
Up to 50							Up to 122
50 - 75							122 - 167
75 - 100	2.7	2.7	0.703	69.8			167 - 212
100 - 125	4.9	7.6	.730	62.3			212 - 257
125 - 150	6.1	13.7	.761	54.4			257 - 302
150 - 175	5.1	18.8	.776	50.9			302 - 347
175 - 200	4.1	22.9	.800	45.4			347 - 392
200 - 225	4.4	27.3	.822	40.6			392 - 437
225 - 250	5.8	33.1	.837	37.6			437 - 482
250 - 275	7.0	40.1	.849	35.2			482 - 527

Vacuum distillation at 40 mm

Up to 200	4.6	4.6	0.870	31.1	41	10	Up to 392
200 - 225	5.7	10.3	.877	29.9	47	30	392 - 437
225 - 250	5.0	15.3	.893	27.0	63	50	437 - 482
250 - 275	5.0	20.3	.902	25.4	100	70	482 - 527
275 - 300	6.1	26.4	.916	23.0	190	95	527 - 572

Carbon residue of residuum, 14.6 percent. Carbon residue of crude, 4.3 percent

APPROXIMATE SUMMARY

	Percent	Sp. gr.	°A.P.I.	Viscosity
Light gasoline	2.7	0.703	69.8	
Total gasoline and naphtha	22.9	.758	55.2	
Kerosene distillate	4.4	.822	40.6	
Gas oil	21.3	.856	33.8	
Nonviscous lubricating distillate	9.3	.880-.902	29.3-25.4	50-100
Medium lubricating distillate	6.2	.902-.917	25.4-22.8	100-200
Viscous lubricating distillate	2.4	.917-.924	22.8-21.6	Above 200
Residuum	29.5	.996	10.6	
Distillation loss	4.0	-	-	

Sample 31044

4970-5457 feet

North Belridge Field
Temblor H. V. fine sandstoneCalifornia
Kern County
27S-20E
M.D.B. & M.

GENERAL CHARACTERISTICS

Specific gravity, 0.813

A.P.I. gravity, 42.6°

Percent sulphur less than 0.1

Color, dark green

Saybolt Universal viscosity at 100°F., 33 seconds

DISTILLATION, BUREAU OF MINES HEMPEL METHOD

Dry distillation

Barometer, 747 mm.

First drop: 31°C. (88°F.)

Temperature, °C.	Per- cent cut	Sum, per- cent	Sp. gr. of cut	°A.P.I. of cut	Viscosity at 100°F.	Cloud test, °F.	Temperature, °F.
Up to 50	1.9	1.9	0.681	76.3			Up to 122
50 - 75	2.6	4.5	.684	75.4			122 - 167
75 - 100	7.2	11.7	.724	63.9			167 - 212
100 - 125	7.7	19.4	.756	55.7			212 - 257
125 - 150	9.0	28.4	.776	50.9			257 - 302
150 - 175	8.5	36.9	.793	46.9			302 - 347
175 - 200	7.3	44.2	.806	44.1			347 - 392
200 - 225	8.5	52.7	.819	41.3			392 - 437
225 - 250	8.9	61.6	.831	38.8			437 - 482
250 - 275	9.3	70.9	.841	36.8			482 - 527

Vacuum distillation at 40 mm

Up to 200	6.0	6.0	0.851	34.8	40	20	Up to 392
200 - 225	7.5	13.5	.855	34.0	46	35	392 - 437
225 - 250	4.7	18.2	.869	31.3	58	60	437 - 482
250 - 275	2.1	20.3	.883	28.8	93	75	482 - 527
275 - 300	2.3	22.6	.897	26.3	180	90	527 - 572

Carbon residue of residuum, 8.3 percent. Carbon residue of crude, 0.5 percent.

APPROXIMATE SUMMARY

	Percent	Sp. gr.	°A.P.I.	Viscosity
Light gasoline	11.7	0.708	68.4	
Total gasoline and naphtha	44.2	.763	54.0	
Kerosene distillate	8.5	.819	41.3	
Gas oil	30.1	.844	36.2	
Nonviscous lubricating distillate	7.6	.860-.884	33.0-28.6	50-100
Medium lubricating distillate	2.5	.884-.900	28.6-25.7	100-200
Viscous lubricating distillate	.6	.900-.903	25.7-25.2	Above 200
Residuum	5.6	.969	14.5	
Distillation loss	.9	—	—	

Sample 37001

CL 11-1
7740-7807 feet

Greeley Field
Miocene Sand

California
Kern County
Sec. 19-29S-26E
M.D.B.M.

GENERAL CHARACTERISTICS

Specific gravity, 0.841
Percent sulphur, 0.28
Saybolt Universal viscosity at 77° F., 46 seconds
Saybolt Universal viscosity at 100° F., 41 seconds

A.P.I. gravity, 36.8°
Pour point, 30° F.
Color, greenish black

DISTILLATION, BUREAU OF MINES HEMPEL METHOD

Dry distillation		Barometer, 749 mm.		First drop: 28° C. (82° F.)			
Temperature, °C.	Per- cent cut	Sum, per cent	Sp. gr. of cut	°A.P.I. of cut	Viscosity at 100° F.	Cloud test, °F.	Temperature °F.
Up to 50	1.7	1.7	0.652	85.5			Up to 122
50 - 75	2.1	3.8	.667	80.6			122 - 167
75 - 100	5.0	8.8	.714	66.7			167 - 212
100 - 125	7.4	16.2	.741	59.5			212 - 257
125 - 150	6.7	22.9	.764	53.7			257 - 302
150 - 175	5.5	28.4	.782	49.5			302 - 347
175 - 200	4.8	33.2	.798	45.8			347 - 392
200 - 225	5.0	38.2	.813	42.6			392 - 437
225 - 250	5.5	43.7	.823	40.4			437 - 482
250 - 275	6.7	50.4	.835	38.0			482 - 527

Vacuum distillation at 40 mm

Up to 200	4.4	4.0	.855	34.0	41	15	Up to 392
200 - 225	5.7	9.7	.861	32.8	47	30	392 - 437
225 - 250	4.3	14.0	.875	30.2	58	55	437 - 482
250 - 275	5.0	19.0	.889	27.7	91	70	482 - 527
275 - 300	5.7	24.7	.905	24.9	170	90	527 - 572

Carbon residue of residuum, 9.5 percent. Carbon residue of crude, 2.2 percent.

APPROXIMATE SUMMARY

	Percent	Sp. Gr.	°A.P.I.	Viscosity
Light gasoline	8.8	0.691	73.3	
Total gasoline and naphtha	33.2	.747	57.9	
Kerosene distillate	10.5	.811	43.0	
Gas oil	15.1	.848	35.4	
Nonviscous lubricating distillate	8.7	.865-.891	32.1-27.3	50-100
Medium lubricating distillate	6.8	.891-.911	27.3-23.8	100-200
Viscous lubricating distillate	.8	.911-.913	23.8-23.5	Above 200
Residuum	22.7	.968	14.7	
Distillation loss	2.2	-	-	

Sample 37002

KCL A Well #A-6-29
7763-8984 at 8383 feet

Ten Section Field
Stevens Sand, Zone V
Miocene

California
Kern County
Sec. 29-30S-26E MDBM

GENERAL CHARACTERISTICS

Specific gravity, 0.836
Percent sulphur, 0.37
Saybolt Universal viscosity at 77°F., 41 seconds
Saybolt Universal viscosity at 100°F., 38 seconds

A.P.I. gravity, 37.8°
Pour point, 10°F.
Color, greenish black

DISTILLATION, BUREAU OF MINES HEMPEL METHOD

Dry distillation		Barometer, 749 mm.		First drop: 27°C. (81°F.)			
Temperature, °C.	Per- cent cut	Sum, per- cent	Sp.gr. of cut	°A.P.I. of cut	Viscosity at 100°F.	Cloud test °F.	Temperature, °F.
Up to 50	4.2	4.2	0.660	82.9			Up to 122
50 - 75	6.0	10.2	.677	77.5			122 - 167
75 - 100	3.9	14.1	.724	63.9			167 - 212
100 - 125	8.0	22.1	.748	57.7			212 - 257
125 - 150	5.8	27.9	.768	52.7			257 - 302
150 - 175	5.3	33.2	.786	48.5			302 - 347
175 - 200	4.4	37.6	.802	44.9			347 - 392
200 - 225	4.3	41.9	.820	41.1			392 - 437
225 - 250	4.9	46.8	.834	38.2			437 - 482
250 - 275	6.4	53.2	.848	35.4			482 - 527

Vacuum distillation at 40 mm

Up to 200	2.7	2.7	.867	31.7	42	15	Up to 392
200 - 225	4.7	7.4	.875	30.2	49	30	392 - 437
225 - 250	4.4	11.8	.892	27.1	62	55	437 - 482
250 - 275	3.8	15.6	.907	24.5	120	70	482 - 527
275 - 300	4.8	20.4	.917	22.8	230	90	527 - 572

Carbon residue of residuum, 12.4 percent. Carbon residue of crude, 2.6 percent.

APPROXIMATE SUMMARY

	Percent	Sp.gr.	°A.P.I.	Viscosity
Light gasoline	14.1	0.685	75.1	
Total gasoline and naphtha	37.6	.730	62.3	
Kerosene distillate	4.3	.820	41.1	
Gas oil	16.8	.851	34.8	
Nonviscous lubricating distillate	6.8	.876-.902	30.0-25.4	50-100
Medium lubricating distillate	4.5	.902-.914	25.4-23.3	100-200
Viscous lubricating distillate	3.6	.914-.922	23.3-22.0	Above 200
Residuum	21.3	.988	11.7	
Distillation loss	5.1	-	-	

Sample 31040

7,276 feet

Kettleman North Dome Field
Sandy Blue ShaleCalifornia
Kings County
21S-17E M.D. B. & M.

GENERAL CHARACTERISTICS

Specific gravity, 0.732
 Percent sulphur less than 0.1
 Saybolt Universal viscosity at 100°F., less than 32 seconds

A.P.I. gravity, 61.8°
 Color, N.P.A. #5

DISTILLATION, BUREAU OF MINES HEMPEL METHOD

Dry distillation Barometer, 745 mm. First drop: 27°C. (81°F.)

Temperature, °C.	Per- cent cut	Sum, per- cent	Sp.gr. of cut	°A.P.I. of cut	Viscosity at 100°F.	Cloud test °F.	Temperature, °F.
Up to 50	6.5	6.5	0.645	87.9			Up to 122
50 - 75	11.7	18.2	.671	79.4			122 - 167
75 - 100	19.9	38.1	.713	65.6			167 - 212
100 - 125	23.0	61.1	.746	58.2			212 - 257
125 - 150	16.9	78.0	.766	53.2			257 - 302
150 - 175	9.7	87.7	.784	49.0			302 - 347
175 - 200*	4.8	92.5	.799	45.6			347 - 392
200 - 225	-	-	-	-			392 - 437
225 - 250	-	-	-	-			437 - 482
250 - 275	-	-	-	-			482 - 527

Carbon residue of residuum, - percent. Carbon residue of crude, - percent.

APPROXIMATE SUMMARY

	Percent	Sp. gr.	°A.P.I.	Viscosity
Light gasoline	38.1	0.691	73.3	-
Total gasoline and naphtha	92.5	.734	61.3	-
Kerosene distillate	Indeterminate			
Gas oil	-	-	-	
Nonviscous lubricating distillate	-	-	-	50-100
Medium lubricating distillate	-	-	-	100-200
Viscous lubricating distillate	-	-	-	Above 200
Residuum	4.2	.841	36.8	
Distillation loss	3.3	-	-	

*Discontinued as slight cracking evidenced at 200°C.

Sample 31041

7065-8323 feet	Kettleman North Dome Field	California
	Temblor	Kings County
	Miocene	21S-17E M. D. B. & M.

GENERAL CHARACTERISTICS

Specific gravity, 0.836	A.P.I. gravity, 37.8°
Percent sulphur, 0.33	Color, brownish black
Saybolt Universal viscosity at 100°F., 39 seconds	

DISTILLATION, BUREAU OF MINES HEMPEL METHOD

Dry distillation		Barometer, 747 mm.		First drop: 27°C. (81°F.)			
Temperature, °C.	Per- cent cut	Sum, per- cent	Sp. gr. of cut	°A.P.I. of cut	Viscosity at 100°F.	Cloud test, °F.	Temperature, °F.
Up to 50	3.0	3.0	0.670	79.7			Up to 122
50 - 75	4.1	7.1	.671	79.4			122 - 167
75 - 100	6.3	13.4	.719	65.3			167 - 212
100 - 125	8.5	21.9	.746	58.2			212 - 257
125 - 150	6.3	28.2	.761	54.4			257 - 302
150 - 175	6.0	34.2	.785	48.8			302 - 347
175 - 200	4.8	39.0	.802	44.9			347 - 392
200 - 225	4.9	43.9	.816	41.9			392 - 437
225 - 250	5.2	49.1	.829	39.2			437 - 482
250 - 275	6.1	55.2	.841	36.8			482 - 527

Vacuum distillation at 40 mm

Up to 200	3.7	3.7	.857	33.6	42	15	Up to 392
200 - 225	4.9	8.6	.862	32.7	47	35	392 - 437
225 - 250	4.7	13.3	.877	29.9	60	55	437 - 482
250 - 275	5.1	18.4	.891	27.3	95	75	482 - 527
275 - 300	6.5	24.9	.907	24.5	210	90	527 - 572

Carbon residue of residuum, 11.2 percent. Carbon residue of crude, 2.1 percent.

APPROXIMATE SUMMARY

	Percent	Sp. gr.	°A.P.I.	Viscosity
Light gasoline	13.4	0.693	72.7	
Total gasoline and naphtha	39.0	.743	58.9	
Kerosene distillate	4.9	.816	41.9	
Gas oil	18.7	.845	36.0	
Nonviscous lubricating distillate	8.8	.865-.892	32.1-27.1	50-100
Medium lubricating distillate	5.0	.892-.905	27.1-24.9	100-200
Viscous lubricating distillate	3.7	.905-.915	24.9-23.1	Above 200
Residuum	18.4	.985	12.2	
Distillation loss	1.5	-	-	

Sample 31042

6,894-7,904 feet	Kettleman North Dome Field	California
	Temblor	Kings County
	Miocene	21S-17E M.D. B. & M.

GENERAL CHARACTERISTICS

Specific gravity, 0.814 ⁱ	A.P.I. gravity, 42.3°
Percent sulphur, 0.22	Color, greenish black
Saybolt Universal viscosity at 100°F., 34 seconds	

DISTILLATION, BUREAU OF MINES HEMPEL METHOD

Dry distillation		Barometer, 745 mm.		First drop: 27°C. (81°F.)			
Temperature, °C.	Per- cent cut	Sum, per- cent	Sp. gr. of cut	°A.P.I. of cut	Viscosity at 100°F.	Cloud test, °F.	Temperature, °F.
Up to 50	3.2	3.2	0.667	80.6			Up to 122
50 - 75	5.5	8.7	.669	80.0			122 - 167
75 - 100	10.6	19.3	.716	66.1			167 - 212
100 - 125	11.0	30.3	.745	58.4			212 - 257
125 - 150	9.3	39.6	.767	53.0			257 - 302
150 - 175	6.7	46.3	.785	48.8			302 - 347
175 - 200	4.6	50.9	.802	44.9			347 - 392
200 - 225	4.4	55.3	.815	42.1			392 - 437
225 - 250	4.0	59.3	.829	39.2			437 - 482
250 - 275	5.4	64.7	.840	37.0			482 - 527

Vacuum distillation at 40 mm

Up to 200	4.8	4.8	.859	33.2	42	20	Up to 392
200 - 225	3.3	8.1	.866	31.9	48	35	392 - 437
225 - 250	3.9	12.0	.882	28.9	62	55	437 - 482
250 - 275	3.8	15.8	.892	27.1	105	75	482 - 527
275 - 300	4.0	19.8	.905	24.9	210	90	527 - 572

Carbon residue of residuum, 9.3 percent. Carbon residue of crude, 1.4 percent.

APPROXIMATE SUMMARY

	Percent	Sp. gr.	°A.P.I.	Viscosity
Light gasoline	19.3	0.694	72.4	
Total gasoline and naphtha	50.9	.740	59.7	
Kerosene distillate	4.4	.815	42.1	
Gas oil	16.5	.846	35.8	
Nonviscous lubricating distillate	6.4	.868-.891	31.5-27.3	50-100
Medium lubricating distillate	4.0	.891-.904	27.3-25.0	100-200
Viscous lubricating distillate	2.3	.904-.911	25.0-23.8	Above 200
Residuum	14.6	.976	13.5	
Distillation loss	.9	-	-	

Sample 37168

8,130-8,557 feet

Kettleman North Dome Field
Temblor 5th Zone SandCalifornia
Kings County
21S-17E M. D. B. & M.

GENERAL CHARACTERISTICS

Specific gravity, 0.855

Percent sulphur, 0.50

Saybolt Universal viscosity at 77°F., 50 seconds

Saybolt Universal viscosity at 100°F., 46 seconds

A.P.I. gravity, 34.0°

Color, brownish black

DISTILLATION, BUREAU OF MINES HEMPEL METHOD

Dry distillation

Barometer, 745 mm.

First drop: 31°C. (88°F.)

Temperature, °C.	Per- cent cut	Sum, per- cent	Sp. gr. of cut	°A.P.I. of cut	Viscosity at 100°F.	Cloud test, °F.	Temperature, °F.
Up to 50	1.9	1.9					Up to 122
50 - 75	2.7	4.6	0.664	81.6			122 - 167
75 - 100	4.8	9.4	.717	65.9			167 - 212
100 - 125	7.0	16.4	.745	53.4			212 - 257
125 - 150	6.1	22.5	.766	53.2			257 - 302
150 - 175	5.1	27.6	.785	48.8			302 - 347
175 - 200	4.3	31.9	.803	44.7			347 - 392
200 - 225	4.6	36.5	.819	41.3			392 - 437
225 - 250	5.2	41.7	.833	38.4			437 - 482
250 - 275	6.5	48.2	.844	36.2			482 - 527

Vacuum distillation at 40 mm

Up to 200	3.8	3.8	0.858	33.4	41	15	Up to 392
200 - 225	5.9	9.7	.866	31.9	48	30	392 - 437
225 - 250	4.5	14.2	.883	28.8	65	50	437 - 482
250 - 275	4.6	18.8	.895	26.6	97	70	482 - 527
275 - 300	5.4	24.2	.908	24.3	200	90	527 - 572

Carbon residue of residuum, 12.4 percent. Carbon residue of crude, 3.2 percent.

APPROXIMATE SUMMARY

	Percent	Sp. gr.	°A.P.I.	Viscosity
Light gasoline	9.4	0.691	73.3	
Total gasoline and naphtha	31.9	.747	57.9	
Kerosene distillate	4.6	.819	41.3	
Gas oil	19.1	.848	35.4	
Nonviscous lubricating distillate	9.2	.868-.895	31.5-26.6	50-100
Medium lubricating distillate	4.9	.895-.908	26.6-24.3	100-200
Viscous lubricating distillate	2.7	.908-915	24.3-23.1	Above 200
Residuum	25.5	.990	11.4	
Distillation loss	2.1	-	-	

Sample 31043

3,516-4,569 feet

Del Rey Field
Del Rey Sand & ShaleCalifornia
Los Angeles County
3S-15W S.B. & M.

GENERAL CHARACTERISTICS

Specific gravity, 0.907
 Percent sulphur, 2.70
 Saybolt Universal viscosity at 100°F., 200 seconds

A.P.I. gravity, 24.5°
 Color, brownish black

DISTILLATION, BUREAU OF MINES HEMPEL METHOD

Dry distillation		Barometer, 755 mm.		First drop: 50°C. (122°F.)			
Temperature, °C.	Per- cent cut	Sum, per- cent	Sp. gr. of cut	°A.P.I. of cut	Viscosity at 100°F.	Cloud test, °F.	Temperature, °F.
Up to 50							Up to 122
50 - 75	2.8	2.8)					122 - 167
75 - 100	1.9	4.7)	0.702	70.1			167 - 212
100 - 125	4.7	9.4	.743	58.9			212 - 257
125 - 150	4.2	13.6	.761	54.4			257 - 302
150 - 175	3.7	17.3	.779	50.1			302 - 347
175 - 200	3.6	20.9	.793	45.8			347 - 392
200 - 225	3.1	24.0	.814	42.3			392 - 437
225 - 250	3.9	27.9	.827	39.6			437 - 482
250 - 275	6.0	33.9	.844	36.2			482 - 527

Vacuum distillation at 40 mm.

Up to 200	1.8	1.8	0.869	31.3	42	15	Up to 392
200 - 225	4.4	6.2	.874	30.4	47	30	392 - 437
225 - 250	4.2	10.4	.885	28.4	58	45	437 - 482
250 - 275	5.2	15.6	.902	25.4	97	60	482 - 527
275 - 300	2.9	18.5	.913	23.5	180	75	527 - 572

Carbon residue of residuum, 14.4 percent. Carbon residue of crude, 6.8 percent

APPROXIMATE SUMMARY

	Percent	Sp. gr.	°A.P.I.	Viscosity
Light gasoline	4.7	0.702	70.1	
Total gasoline and naphtha	20.9	.753	56.4	
Kerosene distillate	3.1	.814	42.3	
Gas oil	15.1	.849	35.2	
Nonviscous lubricating distillate	8.1	.877-.903	29.9-25.2	50-100
Medium lubricating distillate	4.7	.903-.915	25.2-23.1	100-200
Viscous lubricating distillate	0.5	.915-.916	23.1-23.0	Above 200
Residuum	47.0	.993	11.0	
Distillation loss	0.6	-	-	

Sample 37169

3,754-3,924 feet

Fruitvale Field
Chanac, Kernco SandCalifornia
Los Angeles County
29S-27E M.D. B. & M.

GENERAL CHARACTERISTICS

Specific gravity, 0.909

Percent sulphur, 0.60

Saybolt Universal viscosity at 100°F., 140 seconds

Saybolt Universal viscosity at 130°F., 86 seconds

A.P.I. gravity, 24.2°

Color, brownish black

DISTILLATION, BUREAU OF MINES HEMPEL METHOD

Dry distillation

Barometer, 745 mm.

First drop: 90°C. (194°F.)

Temperature, °C.	Per- cent cut	Sum, per- cent	Sp. gr. of cut	°A.P.I. of cut	Viscosity at 100°F.	Cloud test, °F.	Temperature, °F.
Up to 50							Up to 122
50 - 75							122 - 167
75 - 100							167 - 212
100 - 125	1.7	1.7	0.760	54.7			212 - 257
125 - 150	2.2	3.9					257 - 302
150 - 175	2.9	6.8	.799	45.6			302 - 347
175 - 200	3.3	10.1	.825	40.0			347 - 392
200 - 225	3.8	13.9	.848	35.4			392 - 437
225 - 250	5.0	18.9	.862	32.7			437 - 482
250 - 275	7.4	26.3	.868	31.5			482 - 527

Vacuum distillation at 40 mm.

Up to 200	6.3	6.3	0.869	31.3	43	15	Up to 392
200 - 225	8.4	14.7	.874	30.4	49	35	392 - 437
225 - 250	7.5	22.2	.888	27.9	66	55	437 - 482
250 - 275	6.4	28.6	.903	25.2	115	75	482 - 527
275 - 300	8.3	36.9	.913	23.5	240	90	527 - 572

Carbon residue of residuum, 11.8 percent. Carbon residue of crude, 4.3 percent.

APPROXIMATE SUMMARY

	Percent	Sp. gr.	°A.P.I.	Viscosity
Light gasoline				
Total gasoline and naphtha	10.1	0.792	47.2	
Kerosene distillate	-	-	-	
Gas oil	26.9	.865	32.1	
Nonviscous lubricating distillate	12.6	.874-.898	30.4-26.1	50-100
Medium lubricating distillate	7.1	.898-.910	26.1-24.0	100-200
Viscous lubricating distillate	6.5	.910-.919	24.0-22.5	Above 200
Residuum	36.3	.980	12.9	
Distillation loss	.5	-	-	

Sample 37170

3,403-3,472 feet	Fruitvale Field	California
	Etchegoin and Chanac	Los Angeles County
	Contact Sand	29S-27E, M.D. B. & M.

GENERAL CHARACTERISTICS

Specific gravity, 0.958	A.P.I. gravity, 16.2°
Percent sulphur, 0.96	Color, brownish black
Saybolt Universal viscosity at 100°F., 1,690 seconds	
Saybolt Universal viscosity at 130°F., 860 seconds	

DISTILLATION, BUREAU OF MINES HEMPEL METHOD

Dry distillation	Barometer, 745 mm.	First drop: 186°C. (367°F.)
------------------	--------------------	-----------------------------

Temperature, °C.	Per- cent cut	Sum, per- cent	Sp. gr. of cut	°A.P.I. of cut	Viscosity at 100°F.	Cloud test, °F.	Temperature, °F.
Up to 50							Up to 122
50 - 75							122 - 167
75 - 100							167 - 212
100 - 125							212 - 257
125 - 150							257 - 302
150 - 175							302 - 347
175 - 200	0.8	0.8	0.813	42.6			347 - 392
200 - 225	2.4	3.2	.839	37.2			392 - 437
225 - 250	4.0	7.2	.862	32.7			437 - 482
250 - 275	7.3	14.5	.880	29.3			482 - 527

Vacuum distillation at 40 mm

Up to	Per- cent	Sum, per- cent	Sp. gr.	°A.P.I.	Viscosity	Cloud	Up to
Up to 200	4.6	4.6	0.900	25.7	45	Below 5	Up to 392
200 - 225	7.2	11.8	.914	23.3	59	do.	392 - 437
225 - 250	7.3	19.1	.930	20.7	96	do.	437 - 482
250 - 275	7.6	26.7	.946	18.1	240	do.	482 - 527
275 - 300	10.0	36.7	.955	16.7	Over 400	do.	527 - 572

Carbon residue of residuum, 11.9 percent. Carbon residue of crude, 5.7 percent.

APPROXIMATE SUMMARY

	Percent	Sp. gr.	°A.P.I.	Viscosity
Light gasoline				
Total gasoline and naphtha	0.8	0.813	42.6	
Kerosene distillate	-	-	-	
Gas oil	18.2	0.876	30.0	
Nonviscous lubricating distillate	11.2	.906-.930	24.7-20.7	50-100
Medium lubricating distillate	5.1	.930-.941	20.7-18.9	100-200
Viscous lubricating distillate	15.9	.941-.960	18.9-15.9	Above 200
Residuum	47.7	1.007	-	
Distillation loss	1.1	-	-	

R.I. 3362

Sample 37166

3,190-3,618 feet

Wilmington Field
Terminal sand

California
Los Angeles County
5S-13W S.B. B. & M.

GENERAL CHARACTERISTICS

Specific gravity, 0.952

A.P.I. gravity, 17.1°

Percent sulphur, 2.47

Color, brownish black

Saybolt Universal viscosity at 100°F., 1,570 seconds

Saybolt Universal viscosity at 130°F., 950 seconds

DISTILLATION, BUREAU OF MINES HEMPEL METHOD

Dry distillation Barometer, 745 mm. First drop: 99°C. (210°F.)

Temperature, °C.	Per- cent cut	Sum, per- cent	Sp. gr. of cut	°A.P.I. of cut	Viscosity at 100°F.	Cloud test, °F.	Temperature, °F.
Up to 50							Up to 122
50 - 75							122 - 167
75 - 100							167 - 212
100 - 125	1.2	1.2)	0.756	55.7			212 - 257
125 - 150	1.0	2.2)					257 - 302
150 - 175	1.4	3.6)					302 - 347
175 - 200	1.8	5.4)	.795	46.5			347 - 392
200 - 225	2.5	7.9	.829	39.2			392 - 437
225 - 250	4.0	11.9	.850	35.0			437 - 482
250 - 275	7.2	19.1	.868	31.5			482 - 527

Vacuum distillation at 40 mm

Up to 200	2.4	2.4	0.884	28.6	43	Below 5	Up to 392
200 - 225	5.6	8.0	.895	26.6	51	do.	392 - 437
225 - 250	5.3	13.3	.908	24.3	72	do.	437 - 482
250 - 275	5.7	19.0	.922	22.0	135	do.	482 - 527
275 - 300	7.1	26.1	.936	19.7	340	do.	527 - 572

Carbon residue of residuum, 15.5 percent. Carbon residue of crude, 8.2 percent.

APPROXIMATE SUMMARY

	Percent	Sp. gr.	°A.P.I.	Viscosity
Light gasoline				
Total gasoline and naphtha	5.4	0.779	50.1	
Kerosene distillate	-	-	-	
Gas oil	18.5	.864	32.3	
Nonviscous lubricating distillate	8.3	.894-.914	26.8-23.3	50-100
Medium lubricating distillate	5.1	.914-.926	23.3-21.3	100-200
Viscous lubricating distillate	7.9	.926-.944	21.3-18.4	Above 200
Residuum	52.8	1.013	-	
Distillation loss	2.0	-	-	

Sample 37167

3,109-3,905 feet

Wilmington Field
Terminal SandCalifornia
Los Angeles County
5S-13W S.E. B. & M.

GENERAL CHARACTERISTICS

Specific gravity, 0.901

A.P.I. gravity, 25.6°

Percent sulphur, 1.24

Color, brownish black

Saybolt Universal viscosity at 100°F., 110 seconds

Saybolt Universal viscosity at 130°F., 80 seconds

DISTILLATION, BUREAU OF MINES HEMPEL METHOD

Dry distillation		Barometer, 745 mm.		First drop: 32°C. (90°F.)			
Temperature, °C.	Per- cent cut	Sum, per- cent	Sp. gr. of cut	°A.P.I. of cut	Viscosity at 100°F.	Cloud test, °F.	Temperature, °F.
Up to 50	1.1	1.1					Up to 122
50 - 75	1.2	2.3	0.665	81.3			122 - 167
75 - 100	2.9	5.2	.711	67.5			167 - 212
100 - 125	4.5	9.7	.746	58.2			212 - 257
125 - 150	3.7	13.4	.767	53.0			257 - 302
150 - 175	3.5	16.9	.787	48.3			302 - 347
175 - 200	3.5	20.4	.809	43.4			347 - 392
200 - 225	3.9	24.3	.828	39.4			392 - 437
225 - 250	4.9	29.2	.846	35.8			437 - 482
250 - 275	6.3	35.5	.861	32.8			482 - 527

Vacuum distillation at 40 mm

Up to 200	3.7	3.7	0.879	29.5	42	Below 5	Up to 392
200 - 225	5.8	9.5	.891	27.3	52	10	392 - 437
225 - 250	4.8	14.3	.908	24.3	77	25	437 - 482
250 - 275	5.1	19.4	.923	21.8	155	40	482 - 527
275 - 300	6.6	26.0	.936	19.7	400	55	527 - 572

Carbon residue of residuum, 9.0 percent. Carbon residue of crude, 3.3 percent.

APPROXIMATE SUMMARY

	Percent	Sp. gr.	°A.P.I.	Viscosity
Light gasoline	5.2	0.691	73.3	
Total gasoline and naphtha	20.4	0.754	56.2	-
Kerosene distillate	-	-	-	-
Gas oil	20.7	0.857	33.6	-
Nonviscous lubricating distillate	7.8	.888-.912	27.9-23.7	50-100
Medium lubricating distillate	4.5	.912-.925	23.7-21.5	100-200
Viscous lubricating distillate	8.1	.925-.943	21.5-18.6	Above 200
Residuum	36.2	1.007	-	-
Distillation loss	2.3	-	-	-

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REPORT OF INVESTIGATIONS

PROGRESS REPORTS - METALLURGICAL DIVISION

21. STUDIES IN NONFERROUS METALLURGY

COLLECTION OF GOLD BY IRON ABRADED IN GRINDING,
BY S. R. ZIMMERLEY

FLOTATION OF OXIDIZED SILVER-LEAD ORES AS INFLUENCED
BY MODIFIED GRINDING, BY S. R. ZIMMERLEY



REPORT OF INVESTIGATIONS

UNITED STATES DEPARTMENT OF THE INTERIOR - BUREAU OF MINES

PROGRESS REPORTS - METALLURGICAL DIVISION^{1/}

21. Studies in Nonferrous Metallurgy

FOREWORD

In continuing its practice of issuing preliminary reports on subjects of interest to the metallurgical industry so that information regarding its active investigations may be disseminated speedily, the Metallurgical Division presents papers which afford direct evidence that the laboratory phase of the old problem of the effective treatment of certain refractory oxidized silver-lead ores is practically solved.

R. S. Dean,
Chief Engineer,
Metallurgical Division.

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^{1/} The Bureau of Mines will welcome reprinting of this paper, provided the following footnote acknowledgment is used: "Reprinted from Bureau of Mines Report of Investigations 3364."

COLLECTION OF GOLD BY IRON ABRADED IN GRINDING

By S. R. Zimmerley^{1/}

In the course of a flotation investigation, a siliceous gold ore was ground in a ball mill to minus 300-mesh. It was found that the small amount of plus 300-mesh material had a gold assay approximately twice that of the original ore, and that this oversize product also contained much abraded iron. Moreover, the gold assay of the abraded iron, which was separated magnetically, was eight times that of the ore. These facts suggested that some relationship existed between the abnormally high gold content and the presence of abraded iron. It was decided, therefore, to investigate further the association of the gold with the iron abraded from the balls during grinding.

As a preliminary experiment, a sample of siliceous gold ore was ground for 1-1/2 hours in a ball mill and the pulp was divided into two products electromagnetically. The results follow:

Products	Weight, grams	Assay, ounces per ton Au	Distribution, percent Au
Heading	1,156.0	0.154	100.0
Magnetic product	36.3	1.933	38.0
Nonmagnetic tailing	1,120.0	.098	62.0

These data show that the magnetic concentrate that contained the abraded iron was 19.3 times richer than the nonmagnetic material. The magnetic concentrate was not clean, assaying only 14.6 percent iron (a much richer concentrate could have been made by cleaning). The gold content of the concentrate is equivalent to 11.9 ounces per ton of abraded iron, assuming the latter to contain 90 percent Fe.

In this test 38.0 percent of the gold was associated with the abraded iron; however, when the grinding was more prolonged, more of the gold was so associated. After the ore was ground 8 hours in the ball mill, 80.0 percent of the gold was removed when the abraded iron was separated from the pulp with an electromagnet. The gold content of the nonmagnetic tailing, 0.036 ounce per ton, was lower than that of any tailing produced to date by conventional methods of flotation or cyanidation of this refractory ore.

There was a possibility that the association of gold with abraded iron was only an apparent one, and that the magnet may have separated the gold either because the gold itself was faintly magnetic or because it was associated mineralogically with a magnetic mineral. To investigate

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this possibility, the ore was ground for 1-1/2 hours in a porcelain pebble mill, no metallic iron being present. Only 0.9 percent of the total gold was recovered by the magnet. In another test, the ore was ground for 3 hours in the pebble mill, after which iron filings were added to the pulp as a collecting medium for the gold. The magnetic product assayed 0.04 ounce of Au per ton and contained only 0.3 percent of the gold content of the pulp. Since virtually no gold was separated magnetically after grinding in the absence of metallic iron, it is obvious that the abraded iron was the collector for the gold.

A sample of the ore was ground dry by passing it through a pulverizer three times, setting the plates closer after each pass, as is done in preparing a sample for assay. The following results were obtained when the abraded iron was separated magnetically from the ground sample:

Products	Weight, grams	Assay, ounce per ton Au	Distribution, percent Au
Heading	647	0.163	100.0
Magnetic portion	37	.465	16.3
Nonmagnetic portion	610	.145	83.7

The magnetic portion is much richer than the ore itself, and if, on grinding for assaying, this portion were removed, the gold content of the sample would be reduced from actual 0.163 ounce to an apparent 0.145 ounce of Au per ton.

The preceding tests were all made on a hard siliceous ore. Similar tests were made on soft, ocherous, and rougelike ores and also with products rich in sulphides. Although the magnetic portions were richer than the ores, the entire amount of gold removed magnetically from soft ores was small, being only a few percent of the total present. However, the addition of quartz increased the amount of gold collected, as shown by the following experiments: On grinding 1,000 grams of a rougelike ore for 5-1/2 hours, only 1.7 percent of the gold was recovered in the magnetic product. In a similar test, wherein 500 grams of quartz was mixed with 500 grams of the ore, 11.5 percent of the gold was recovered. The magnetic concentrates assayed, respectively, 10.25 ounces and 9.4 ounces of gold per ton.

The abraded iron, bearing the gold, floated readily from freshly ground pulp, as is shown in the following experiments, in which Pentasol xanthate and Du Pont B-23 were used as reagents. A sample of ore was ground for 4 hours and floated. The tailing from flotation was carefully gone over with an electromagnet to remove all abraded iron. The following results were obtained:

Products	Weight, grams	Assay, ounces per ton Au	Distribution, percent Au
Headings	1,000	0.253	100.0
Flotation concentrate	51.0	4.710	94.5
Magnetic material from tailing.	7.5	.180	0.6
Tailing.....	969.0	.013	4.9

In this test, only 0.6 percent of the gold was found with the abraded iron in the tailing. In two other tests with shorter grinding periods this loss was found to be only 0.3 percent and 0.4 percent of the gold. It does not follow, necessarily, that the gold that floated was associated with the abraded iron, because free gold itself is floatable if clean and fine. However, in another test on the same ore, in which the grinding period was 2-1/2 hours instead of 4 hours, the abraded iron was removed magnetically from the pulp and the magnetic product was found to contain 46.2 percent of the gold. It is evident, therefore, that much of the gold in the concentrate was associated with the abraded iron and that virtually none so associated was lost in the tailing.

Another type of ore from this same mine that contained much clay was also tested. In this case the abraded iron also floated reasonably well, only 3.4 percent of the gold in the ore being removed from the flotation tailing by the magnet.

SUMMARY

1. Native gold is definitely collected on the metallic iron surfaces during grinding in a ball mill. This iron, when abraded and subsequently collected either magnetically or by flotation, will account for most of the gold present if grinding is continued long enough and if the ore is hard and siliceous.

2. If the gangue material is softer than the iron balls, the amount of gold collected by abraded iron is very small.

3. Grinding a siliceous gold ore in a pulverizer is accompanied by some segregation of the gold with the abraded iron.

4. The mechanism of the collection of gold by metallic iron appears to be one by which the malleable gold is smeared on the roughened surfaces of the grinding media; as grinding progresses, small particles of iron together with the adherent gold are removed by abrasion by the hard gangue material.

FLOTATION OF OXIDIZED SILVER-LEAD ORES AS INFLUENCED BY MODIFIED GRINDING

By S. R. Zimmerley^{1/}

INTRODUCTION

In the previous paper in this report, it was shown that when a siliceous gold ore is ground for a long time in a ball mill, much abraded iron is found in the pulp. Most of the gold in the ore was associated with the abraded iron^{2/} when it was removed with an electromagnet. A siliceous, oxidized, lead-silver ore was ground to determine its behavior under similar conditions. Very little metallic iron was found upon magnetic separation; instead, there was present a surprisingly large amount of heavy sponge lead in the settled pulp and a corresponding amount of ferrous iron in the supernatant liquid. From the presence of these products it was evident that the oxidized lead minerals had reacted with the iron balls to form sponge lead. Thus, for anglesite



As the solubility of oxidized lead minerals in water is small, this extensive formation of sponge lead during grinding was not anticipated. However, this discovery, together with the fact that the sponge lead and the silver could be floated readily, suggested a plan of attack for the oxidized silver-lead-ore problem.

OXIDIZED SILVER-LEAD-ORE PROBLEM

The treatment of the low-grade, oxidized, silver-lead ores has long been a problem in Utah. Two plants^{3/} were built for the treatment of Tintic ores by chloridizing roasting and leaching. Later, a plant was built to utilize oxide volatilization.^{4/} Finally, a large mill and several smaller ones were built for the flotation of these ores.^{5/} None

^{1/} Associate metallurgist, Metallurgical Division, Bureau of Mines.

^{2/} Dean, R. S., Annual Report of the Metallurgical Division, Fiscal Year 1936. R. I. 3331, Bureau of Mines, pp. 36-37.

^{3/} Higgins, Will C., Plant and Practice of the Tintic Milling Co.: The Salt Lake Mining Review, vol. 18, Nov. 30, 1916, p. 17.

Allen, H. P., and Madge, W. C., Chloridizing Mill of the Standard Reduction Co.: Trans. Am. Inst. Min. and Met. Eng., vol. 73, 1926, pp. 317-341.

^{4/} Wigton, G. H., Chief Consolidated Volatilization Process and Mill: Trans. Am. Inst. Min. and Met. Eng., vol. 73, 1926, pp. 200-209.

^{5/} Bean, J. J., Milling Methods of the Oxide Concentrator of the International Smelting Co., Tooele, Utah: Inf. Circ. 6759, Bureau of Mines, 1933, 8 pp. Wigton, G. H., Milling Methods and Costs at the Concentrator of the Chief Consolidated Mining Company, Eureka, Utah: Inf. Circ. 6320, Bureau of Mines, 1930, 18 pp.

of these plants is in operation at present, although ores are still being mined.

The history of standard flotation of these oxidized ores has shown certain inherent difficulties, and the losses in the tailings have been great. Many reagents were used, and their consumption has been large. The oxidized ores, per se, are complex and variable. The lead occurs as galena, anglesite, plumbojarosite, cerussite, and in small amounts as bindheimite, wulfenite, and other minerals. The relative amounts of the sulphide and oxidized minerals vary considerably in the ore, as do the amounts of anglesite and cerussite in the oxidized portion. The occurrence of the silver is variable, also, and is associated with lead minerals to a considerable extent. The gangue portion, although chiefly hard siliceous materials, also contains oxidized iron and products of rock decomposition.

The chance of finding an effective combination of selective reagents that will float such a large variety of minerals does not appear hopeful. The change in the proportions of the various component minerals in the feed from day to day adds further complications. With this in mind, the need for simplification is evident.

If the oxidized lead minerals can be converted to sponge lead during grinding, the process then becomes one of floating a simple constituent, the metallics. Silver, rather than lead, is the most valuable constituent of the ores; moreover, the recovery of this element by standard flotation practice is particularly inadequate. The recovery of silver is also greatly increased by this process of metallizing and floating, as the data in the text will show. This paper will give, first, the results obtained on minable ores and, later, the more fundamental factors governing the formation of sponge, including the rates of metallizing various pure minerals. As this is a progress report, the work is incomplete and many factors need to be investigated further. A study has been made of the factors affecting the conversion of oxidized lead minerals to sponge metal during grinding and of the flotation of the lead and silver from the ground product.

Description of Ores Used

Two Tintic ores, which were particularly suited to the investigation, were available. Both were being shipped to smelters without being concentrated; considerable work had been done on each by other investigators, and neither ore was found to be amenable to treatment by orthodox methods of flotation. One ore, the Tintic Standard, is of the partly oxidized type in which both sulphides and oxidized minerals occur. The other, the Chief Consolidated, is virtually completely oxidized, the sulphide content being almost nil. If these two ores can be treated successfully, obviously other similar types, intermediate in degree of oxidation, will be amenable to treatment.

CHIEF CONSOLIDATED ORE

The chemical analysis of this almost completely oxidized siliceous ore follows, in percent: Pb, 8.1; Fe, 7.2; Cu, 0.10; Zn, 2.2; CaO, 1.6; total S, 1.34; SiO₂, 57.0; Al₂O₃, 2.3; Ba, 0.9; As, 0.31; Sb, 0.06; and Mn, 0.21. The precious-metal content, in ounces per ton, is Ag, 11.0, and Au, 0.05.

The following results (table 1) were obtained by another investigator experienced in the treatment of this ore. In test B, a sulphidizing procedure was used; in test A, C-C reagent^{6/} and a mixture of xanthates in large quantity were substituted for the sulphidizing reagents.

TABLE 1. - Results of two flotation tests made by outside investigators

	Percent weight	Assay, ounces per ton		Assay, percent	Distribution, percent		
		Ag	Au	Pb	Ag	Au	Pb
<u>Test A:</u>							
Heading-----	100.0	11.5	0.07	15.4	100.0	100.0	100.0
Concentrate--	33.2	22.9	.10	39.0	65.6	50.0	83.2
Tailing-----	66.8	5.9	.05	3.8	34.4	50.0	16.8
<u>Test B:</u>							
Heading-----	100.0	13.2	.07	9.8	100.0	100.0	100.0
Concentrate--	29.9	28.9	.12	27.2	65.5	51.0	83.3
Tailing-----	70.1	6.5	.05	2.3	34.5	49.0	16.7

It will be noted that the recovery of the lead was fair, but the silver content of the tailing was high. This silver was thought to be either in solid solution in the silica or so finely disseminated therein that it could not be freed and recovered by a flotation process.

The better recovery of silver should be noted in our succeeding experiments, wherein the lead minerals are altered by continued grinding with iron balls and the resultant sponge is floated. The gradual decrease in silver content of the tailing with improved procedure is of particular interest when it is borne in mind that 5 to 6 ounces per ton was considered by other experimenters as the lower limit attainable by flotation.

Preliminary Experiments

In the preliminary tests, 1,000 grams of ore and 600 ml of tap water were ground four hours in the ball mill. Tailings were produced by panning and flotation, assaying from 4.25 ounces per ton (with a high ratio of concentration) to 3.7 ounces per ton (at a lower ratio). The recovery of lead was poor. Finally, by grinding in four stages of 1 hour

^{6/} Wigton, G. H., Reference 5.

each, floating between stages, and cleaning the concentrates, a tailing was produced that assayed 2.75 ounces of silver per ton. Table 2 gives the results.

TABLE 2. - Results of preliminary flotation test

Products	Weight, grams	Assay, ounces per ton	Distribution Ag, percent
Heading.....	1,000	11.50	100.0
Concentrate.....	196	44.35	74.0
Cleaner tailing.....	212	6.80	12.3
Rougher tailing.....	615	2.75	13.7

To improve the recovery of lead, 100 grams of salt (NaCl) was added to the pulp before grinding for 4 hours. The ground pulp was panned and then floated, 0.23 pound of Pentasol xanthate per ton and two drops of pine oil being used. The flotation concentrate was cleaned without additional reagents. Table 3 gives the results.

TABLE 3. - Results of flotation test with brine

Products	Weight, grams	Assay		Distribution, Percent	
		Ounces Ag	Percent Pb	Ag	Pb
Heading.....	1,000	11.0	8.1	100.0	100.0
Panned concentrate.....	55	42.8	48.0	20.8	31.4
Flotation concentrate...	87	60.5	28.5	46.4	29.5
Cleaner tailing.....	332	7.3	6.2	21.4	24.5
Rougher tailing.....	559	2.3	2.2	11.4	14.6

By using brine in grinding, the lead was separated without sulphidizing and the silver in the tailing was reduced to 2.3 ounces per ton. In most of the subsequent tests grinding has been done in brine, as improved results were obtained and the cost of salt in the vicinity of Great Salt Lake is low.

In test 169, the ore was ground for 1/2 hour in tap water and some of the lead minerals were removed by sulphidizing and floating. An emulsion of Ninol 443 and oil was used for flotation. The oil emulsion was not well adapted to grinding subsequently in brine; despite 8 hours grinding, the final tailing after flotation assayed 4.8 percent lead. The silver in the tailing assayed only 2.3 ounces, and middling products were avoided.

Another test, 170, was made similar to 169, but the oil emulsion was omitted. Pentasol xanthate and pine oil were substituted as reagents for

the three flotation stages noted in table 4.

TABLE 4. - Results of flotation test 170

Products	Weight, grams	Assay, ounces per ton		Assay, percent	Distribution, percent		
		Ag	Au	Pb	Ag	Au	Pb
Headings.....	1,000	11.5	0.053	7.9	100.0	100.0	100.0
Concentrate from sulphidizing.....	108	54.3	.170	52.8	51.1	34.7	72.1
Concentrate after first 4-hour grind.....	95	34.0	.170	6.6	28.2	30.6	7.9
Concentrate after second 4-hour grind.....	56	17.4	.110	4.7	8.5	11.7	3.2
Tailing.....	807	1.75	.015	1.65	12.2	23.0	16.8
All concentrates.....	259	38.9	.160	25.4	87.8	77.0	83.2

The results show that tailing was produced containing only 1.75 ounces of silver per ton, and the recoveries of the other metals were good for this type of ore. It will be noted that the operation of sulphidizing precedes the long-grinding stage. Sulphidizing after the long-grinding stage has been tried, but the results were less satisfactory than when the bulk of the easily floatable lead had been removed first, as in this experiment.

Use of Acidic Grinding Circuits

In a preliminary test, sulphuric acid was added to the ore and brine during grinding. The flotation concentrate assayed 23⁴ ounces of silver per ton, 64 percent being recovered. The grinding time was only 1-1/2 hours. Experimental difficulties were encountered when opening and closing the mill to add acid at short intervals. To avoid this trouble, the ore was first ground in tap water and transferred from the mill to a porcelain dish containing salt. Then sulphuric acid was added gradually as required to maintain a slight acidity in the pulp. An amount of sulphuric acid equivalent to 122 pounds per ton was added in this manner. The pulp was then ground 1 hour in the ball mill. The pulp was neutral after grinding and more acid (28 pounds per ton) was required to make it temporarily acidic to methyl orange just before floating. Pentasol xanthate and pine oil were used as reagents. The concentrate was cleaned and the tailings were combined, thickened, made acidic by the addition of 4 pounds of H₂SO₄ per ton, and ground 2 hours more in the ball mill. The pulp was floated as in the first stage, except that the concentrate was not cleaned. Table 5 gives the results.

TABLE 5. - Results of flotation test 183, acid circuit

Products	Weight, grams	Assay,		Distribution, percent	
		Ounces Ag	Percent Pb	Ag	Pb
Heading.....	1,000	13.00	8.1	100.0	100.0
Concentrate after grinding 1 hour.	31	295.50	12.6	70.3	4.8
Concentrate after grinding 2 hours more.....	22	108.40	38.3	18.2	10.4
Tailing.....	945	1.58	7.3	11.5	84.8

The silver content of the combined concentrates, 217.8 ounces per ton, was high, and that of the tailing was the lowest yet produced, 1.58 ounces per ton. Assays of a concentrate panned from the final tailing showed no concentration of lead, hence the high lead content of the tailing was not due to the formation of lead pellets or the failure of coarse sponge to float. The high consumption of acid, 164 pounds per ton, was not encouraging; but the tests show what might be expected when this type of ore is treated first with acid. The excellent results with reference to silver also suggest that this process may be adapted to an oxidized silver ore when the lead recovery is unimportant and acid is cheap, or when the ore consumes little acid.

To investigate the effect of adding acid gradually during grinding, with the pH controlled within definite limits, an open mill was used. Samples for spot testing were removed readily and the acid was added as often as required. When the pH exceeded about 4.8, more acid was added. Acid was added during the first half hour, then grinding was continued in the basic pulp for 2 hours more. On floating, 77.9 percent of the silver and 72.6 percent of the lead were recovered.

In test 192, the easily floatable lead minerals were first removed by sulphidizing and floating. The tailing was thickened and put into the open mill used for the previous test; however, for this test the pulp was made up with tap water instead of brine. After 1-1/2 hours of grinding, 105 pounds of H_2SO_4 per ton had been added; grinding was continued for another hour. For flotation, an emulsion of ethyl dixanthogen and X-1 worked well as a reagent combination for all stages, an additional amount of X-1 being added as required for proper frothing. The tailing assayed 2.85 ounces Ag per ton and 2.35 percent Pb, with corresponding recoveries of 82.6 and 63.9 percent. These results are fair, considering that the ore was ground for only 2 hours in a pulp containing no salt.

Grinding in Hot Brine

In this test, 196, the ore was ground in hot brine to accelerate the formation of sponge. First, the ore was ground 1/2 hour in tap water at room temperature and then sulphidized and floated. Salt was added to the thickened tailing and grinding was carried out in hot brine, average

temperature 79.5°C., for 1 hour. The pulp was discharged from the closed mill and diluted for flotation. Amyl dixanthogen X-1. emulsion had been used in the preliminary stage, and there was enough reagent remaining for the beginning of flotation in the second stage. Later, 6 pounds of H_2SO_4 per ton, 2 pounds of $CuSO_4 \cdot 5H_2O$, and a small amount of the emulsion were added. The froth, at first black, finally became white and barren, the end of the flotation step being sharply defined. The assays of the products follow in table 6.

TABLE 6. - Results of flotation test 196,
hot-brine grind

Products	Weight, grams	Assay, ounces per ton		Assay, percent	Distribution, percent		
		Ag	Au	Pb	Ag	Au	Pb
Heading.....	1,000	11.60	0.056	7.6	100.0	100.0	100.0
Preliminary concentrate	156	46.60	.150	37.5	62.6	41.9	76.9
Concentrate after 1 hour grinding.....	70	37.50	.170	7.7	22.6	21.3	7.1
Tailing.....	810	2.13	.025	1.5	14.8	36.8	16.0
Ratio of concentration 4.4 to 1.							

The tailing from this test was panned before drying, and table 7 gives the results.

TABLE 7. - Results of flotation test 196-A

Products	Weight, grams	Assay, Ounces per ton Percent		Percent of total in headings	
		Ag	Pb	Ag	Pb
Concentrate panned from tailing.....	10	4.75	2.30	0.4	0.3
Remainder of tailing	800	2.10	1.50	14.4	15.7
Total tailing.....	810	2.13	1.51	14.8	16.0

From tables 6 and 7 the following facts were shown: 1. The tailing assays only 1.5 percent Pb and 2.13 ounces Ag per ton and contains almost no heavy metallic lead or silver. 2. The ratio of concentration is relatively high. 3. This has been accomplished with only 1-1/2 hours total grinding. 4. More of the gold is recovered than by standard flotation. 5. The metal values recovered in the second stage permitted an additional cost of about \$2.75 per ton of ore for this stage.

The next procedure tried was somewhat similar to the foregoing; this time, however, the sulphidizing step was omitted. The ore was ground 1/2

hour in brine at room temperature and floated, amyl dixanthogen X-1 emulsion; 3 pounds of sodium silicate per ton and 11 pounds of H_2SO_4 were used as reagents. The thickened tailing was ground in hot brine, average temperature $72^{\circ}C.$, for 1-1/2 hours. In the cell there were added 20 pounds of H_2SO_4 per ton, 2 pounds of $CuSO_4$, and 2 ml of diluted emulsion equal to 0.03 pound amyl dixanthogen and 0.01 pound of X-1 per ton. As the first part of the concentrate, 2(a), appeared to be very clean, it was collected separately from the rest of the concentrate, 2(b). No concentrates were cleaned. (See table 8.)

TABLE 8. - Results of flotation test 208,
hot-brine grind

Products	Weight, grams	Assay,		Distribution, percent	
		Ounces per ton Ag	Percent Pb	Ag	Pb
Heading.....	1,000	11.4	7.7	100.0	100.0
Concentrate 1.....	48	124.9	9.6	52.5	6.1
Concentrate 2(a).....	63	33.5	58.2	18.5	47.5
Concentrate 2(b).....	53	25.8	20.2	12.0	20.2
Tailing.....	883	2.2	2.3	17.0	26.2

In this test, 83 percent of the silver and 73.8 percent of the lead were recovered without sulphidizing. The concentrate 2(a) assayed 58.2 percent lead, a rich concentrate for the flotation of oxidized lead ore. The dixanthogen emulsion seems well adapted for the sponge flotation. However, the preliminary separation of the lead was disappointing. Better results were obtained in the preliminary separation in another test, 202, (table 9), employing the same grinding period but using pentasol xanthate and pentasol alcohol in flotation.

TABLE 9. - Results of flotation test 202

Product	Weight, grams	Assay,		Distribution, percent	
		Ounces per ton Ag	Percent Pb	Ag	Pb
Concentrate 1.....	62	100.25	28.3	24.9	56.5

A substantial amount of the lead (56.5 percent) was removed as a fair-grade product without sulphidizing, and the overall recovery would probably have been better than for 208 had the same subsequent treatment been given.

It is thought that with a little more work on this ore a better combination of reagents will be found, so that a substantial portion of the lead can be removed as in test 202, without sulphidizing in the preliminary flotation. Then, grinding in hot brine for a short period and

Floating will be sufficient to make a tailing as satisfactory as that produced by the longer grinding and more complex treatment of test 170 and without the consumption of as large amounts of acid as in test 183.

TINTIC STANDARD ORE

The siliceous ore is partly oxidized; however, it contains considerable pyrite, and much of the silver is associated with the sulphide minerals. The chemical analysis follows, in percent: Pb, 4.55; Fe, 11.9; Cu, 0.32; Zn, 0.90; S, 7.00; SiO₂, 68.0; Al₂O₃, 1.50; Ba, 2.47; CaO, 0.43; As, 0.43; Sb, 0.11; Mn, 0.06. The precious-metal content is 21.9 ounces of Ag and 0.051 ounce of Au per ton. The mineralogical occurrence of the silver is described by R. E. Head.⁷

The results shown in table 10, obtained by another investigator, are typical of the response of this ore to conventional flotation methods.

TABLE 10. - Results of flotation tests
by outside investigator

Product	Weight, percent	Assay, percent				Assay, oz. per ton	
		Pb	Fe	Cu	Insol.	Ag	Au
Heading.....	100.0	4.6	11.2	0.34	68.1	22.00	.045
Concentrate...	15.4	6.4	33.2	1.52	12.3	100.65	.172
Middling.....	9.3	6.4	11.8	.26	64.1	18.80	.050
Tailing.....	75.3	3.8	7.6	.10	77.0	5.70	.017

The tailing was found to contain virtually no sulphide minerals when examined microscopically. Thus, the tailing represented the losses in the oxidized portion of the ore, and it will be noted that it contained nearly all of the lead and assayed 5.7 ounces per ton in silver.

A portion of the tailing described in table 10 was obtained from the investigator. One thousand grams of the tailing was corrected for alkalinity, using 10 pounds of H₂SO₄ per ton; it was then ground with 650 ml of tap water and 150 grams of NaCl for 3-1/2 hours in a ball mill. The concentrate was next floated, using 0.1 pound of pentasol xanthate per ton of ore and a drop of pine oil. As the quantity of reagent was too great for this finely divided pulp, the concentrate was cleaned without additional reagents. The results shown in table 11 were obtained:

⁷/ Head, R. E., Mineralogical Characteristics Affecting the Concentration of a Semi-oxidized Lead-silver Ore: (In preparation).

TABLE 11. - Results of flotation test 143, brine grind

Product	Weight, grams	Assay, ounces per ton		Assay, percent Pb	Distribution percent		
		Ag	Au		Ag	Au	Pb
Heading.....	1,000	5.60	0.019	3.9	100.0	100.0	100.0
Concentrate.....	28	93.25	.250	5.3	46.3	36.1	3.7
Cleaner tailing..	224	5.00	.020	5.3	19.9	23.1	29.6
Tailing.....	792	2.40	.010	3.4	33.8	40.8	66.7

These data show that the silver content of the original tailing has been reduced from 5.6 ounces to 2.4 ounces per ton by virtue of the grinding process. Better results could be expected from the direct treatment of the ore than from the retreatment of the above tailing.

Preliminary Experiments

The ore instead of the tailing was used in all the following tests: Flotation of the sulphides, after a short grinding period of 1/2 hour, has been the usual procedure. This has permitted the sulphide portion, rich in silver, to be removed before being ground so fine as to render recovery by flotation difficult. After the sulphides have been removed, the thickened tailing has been ground for various periods under varied conditions of mill speed, pulp composition, and temperature. A closed mill has been used for grinding, unless otherwise noted.

The following is a description of one of the earlier tests in which the Tintic Standard ore was used. One thousand grams of ore and 600 ml of tap water were ground 1/2 hour in the ball mill. The sulphides were floated, using two pounds of sodium silicate per ton, 0.1 pound of potassium ethyl xanthate, and a drop of pine oil. The sulphide concentrate was dried. The tailing was thickened, dewatered, mixed with 150 grams of salt and 10 pounds of H₂SO₄ per ton, and then ground 3-1/2 hours in the ball mill. This pulp was floated at about 30 percent solids, 0.2 pound of pentasol xanthate per ton and a drop of pine oil being used as reagents. The rougher tailing was dried. The concentrate was refloated without reagents producing concentrate 2 and a cleaner tailing. Table 12 gives the results.

TABLE 12. - Results of flotation test 144,
brine-acid grind

Product	Weight, grams	Assay,		Distribution, percent	
		Ounce per ton Ag	Percent Pb	Ag	Pb
Heading.....	1,000	20.3	4.2	100.0	100.0
Sulphide concentrate..	150	94.8	6.5	66.7	21.2
Concentrate 2.....	51	84.9	35.4	20.2	40.6
Cleaner tailing.....	106	5.4	3.1	4.0	7.4
Rougher tailing.....	745	2.6	1.8	9.1	30.1

A comparison of these results with those in Table 10 shows that the amount of both lead and silver in the tailing has been greatly reduced. Concentrate 2 represents the gain by protracted grinding brine, 20 percent additional silver and 40 percent of the lead being saved. It is rich enough to ship to a lead smelter, thereby bringing a return for the lead.

Test 147 was similar to the foregoing, except that 3 pounds of Na_2S per ton was added to the cell after the preliminary flotation of the sulphides. The concentrate from sulphidizing was later combined with the lead concentrates. The thickened tailing was ground 4 hours in brine and then floated, as in test 144.

TABLE 13. - Results of flotation test 147,
with sodium sulphide

Products	Weight, grams	Assay,		Distribution, percent	
		Ounce per ton	Percent	Ag	Pb
		Ag	Pb		
Heading.....	1,000	21.8	4.6	100.0	100.0
Sulphide concentrate..	148	112.6	6.7	75.2	21.6
Lead concentrate.....	118	33.1	15.9	17.7	40.7
Tailing.....	751	2.1	2.1	7.1	37.7

The products of this test have been combined in table 13 so that there are no intermediate products, only concentrates and tailing.

In another test, otherwise similar to 144, the pulp was ground for 8 hours in brine. The results are given in table 14.

TABLE 14. - Results of flotation test 168,
brine grind

Products	Weight, grams	Assay,		Distribution, percent	
		Ounce per ton	Percent	Ag	Pb
		Ag	Pb		
Heading.....	1,000	22.7	4.6	100.0	100.0
Sulphide concentrate..	132	104.4	4.6	61.2	13.0
Lead concentrate.....	122	56.4	22.1	30.7	57.8
Tailing.....	804	2.3	1.7	8.1	29.2

Although the tailing assays about the same as 144, more of the lead, 57.8 percent, has been recovered in the lead concentrate; moreover, the ratio of concentration is improved.

Improved Procedure

Test 171 was made to determine the degree of separation attainable by

combining the favorable features of previous tests. After grinding the ore 1/2 hour, the sulphides were floated. Next, 2 pounds of Na_2S per ton was added to the cell and the sulphidized concentrate was removed. Salt was added to the thickened tailings and this pulp was ground 4 hours. A concentrate was made using 0.1 pound of Pentasol xanthate per ton and two drops of Pentasol alcohol as reagents. This concentrate was cleaned without additional reagents. The bulk sulphide concentrate was cleaned later, dichromate being used to depress the lead. The depressed portion of the bulk sulphide concentrate, the concentrate from sulphidizing, and the sponge concentrate are all combined in the tabulation as lead concentrates. Likewise, all tailings are represented as one product. Table 15 lists the results.

TABLE 15. - Results of flotation test 171,
with sodium sulphide and brine grind

Product	Weight, grams	Assay, ounces per ton		Assay, percent Pb	Distribution, percent		
		Ag	Au		Ag	Au	Pb
Heading.....	1,000	21.9	0.051	4.54	100.0	100.0	100.0
Pyrite concentrate..	103	115.4	.195	4.30	54.3	39.1	9.8
Lead concentrate...	191	61.4	.158	19.80	39.3	43.1	61.2
Tailing.....	825	1.7	.011	1.60	6.4	17.8	29.0

The tailing contained 91.7 percent of the insoluble content of the original ore.

Increased Return from Concentrating

The net value of these concentrates has been estimated from current smelter schedules. These computations show the concentrate from 1 ton of untreated ore to be worth about \$3 more than the untreated ore. At present the untreated ore would net about \$14 per ton. These figures allow considerable leeway and justify more than the usual expenses for milling. Concentration by conventional flotation, as typified by table 10, is not economical at all - the net value of the concentrate does not equal the net return from the untreated ore.

Depressing Lead from Bulk-sulphide Concentrate

It is desirable to separate the small amount of galena from the large amount of pyrite in the sulphide concentrate, later combining it with the lead concentrate. The pyrite, the major constituent, is active, and a separation based on depressing the pyrite with zinc sulphate, lime, and cyanide was not successful. However, if a small amount of potassium chromate and sulphuric acid is added to the bulk concentrate before re-floating, the amount of lead in the pyrite concentrate can be reduced. The following examples (table 16) are taken from several tests. The bulk

sulphide concentrate average around the 6.4 percent Pb, as given in table 10.

TABLE 16. - Assay of cleaner sulphide concentrate

Products	Weight, grams	Assay, percent Pb	Assay, ounces per ton Ag	Assay, percent Fe
<u>Test 176:</u>				
Pyrite concentrate....	49	1.45	88.0	40.0
Recleaner tailing.....	58	6.55	123.5	36.3
Cleaner tailing.....	15	9.65	145.2	16.1
<u>Test 218:</u>				
Pyrite concentrate....	107	3.65	85.1	39.7
Recleaner tailing.....	15	7.60	228.4	31.0
Cleaner tailing.....	44	14.70	87.0	16.2
<u>Test 219:</u>				
Pyrite concentrate....	104	3.20	69.8	39.5
Cleaner tailing.....	75	13.20	134.1	17.7

From these data it can be seen that pyrite concentrates, definitely impoverished in lead, can be made if such products are desired for shipment to a copper smelter for recovery of the silver. The proportion of pyrite concentrate made is largely a matter of the period for flotation and can be controlled. Thus, the proportion of the silver production going to the copper or lead smelter could be adjusted to take advantage of the schedules offering the greater return.

In the following test the sulphides were first removed by using 0.12 pound of potassium ethyl xanthate per ton and pine oil. The bulk concentrate was then put into the cell and conditioned 10 minutes with 0.25 pound of K_2CrO_4 per ton and 3 pounds of H_2SO_4 , and then floated without additional frother or collector. The concentrate was cleaned again in the same manner. After removing the bulk sulphide concentrate, the same general procedure was followed in treating the remaining pulp, as described for test 171. This consisted of sulphidizing, grinding 4 hours in brine, and floating, and an additional step - grinding 2 hours more in brine acidulated acetic acid and making another concentrate. The depressed lead from the bulk sulphide separation, the sulphidizing concentrate, and the two sponge concentrates are all combined in table 17 as "lead concentrates."

TABLE 17. - Results of flotation test 176,
as described

Products	Weight, grams	Assay,		Distribution, percent	
		Ounces per ton Ag	Percent Pb	Ag	Pb
Heading.....	2,000	22.00	4.30	100.0	100.0
Pyrite concentrate...	98	88.00	1.45	19.6	1.5
Lead concentrate.....	382	86.70	20.20	75.4	80.1
Tailing.....	1,564	1.43	1.13	5.0	18.4

The procedure in this test was involved and lengthy; however, the results show what can be accomplished on concentrating this ore. The tailing was the cleanest produced, assaying 1.43 ounces of Ag per ton and 1.13 percent of Pb. Eighty percent of the lead was recovered as a salable concentrate and 95.0 percent of the silver was recovered in the concentrates. A definiteness is afforded in the results by the absence of middling products.

The problem at this stage is to shorten the procedure without seriously altering the metallurgical results. To accelerate the reaction, adding H_2SO_4 during grinding, was tried, and finally grinding was carried out in heated brine solutions.

Acidic Grinding Circuits

The tests wherein H_2SO_4 was added were similar to those described for the Chief Consolidated ore. The Tintic Standard ore was treated with H_2SO_4 in a porcelain dish and in other tests ground in a large open mill, acid being added as required and methyl orange being used as an indicator. This ore, after being ground for 1/2 hour, transferred to a porcelain dish, and then acidified, required 25 pounds of H_2SO_4 per ton to produce temporary acidity: thereafter only 10 pounds per ton was required with occasional stirring to maintain permanent acidity to methyl orange for 48 hours. However, when the acid was added during grinding, despite lack of great acidity at any time, 61 pounds of H_2SO_4 per ton was consumed in 1 hour, and the pulp was basic to methyl orange at the end of this period. (This is another example of how grinding greatly increases the rate of reaction.) The results after grinding the acid treated ore and floating were not encouraging. The grinding period had been reduced to 2-1/2 hours, but the rougher tailing assayed 2.4 ounces per ton Ag and 2.6 percent Pb; the combined tailings, 3.5 ounces of Ag per ton and 2.3 percent Pb.

Grinding in Hot Brine

A 1,000-gram sample of the ore was ground with 600 ml of water and 200 grams of salt for 1 hour at an average temperature of $61^{\circ}C$. and floated. Without the benefit of a preliminary separation, 86.4 percent of the silver and 50 percent of the lead were recovered. An emulsion of amyl dixanthogen and water prepared with Emulsol X-1 was used; X-1 also acted as a frother; it is stable in an acidic circuit. A clean end-point was obtained by adding 5 pounds of H_2SO_4 per ton near the end of the flotation operation.

In test 195, the ore was ground 1/2 hour in water and the sulphides were floated. The reagents were 3 pounds of sodium silicate per ton and the amyl dixanthogen, X-1 emulsion. The thickened tailing was ground 1 hour in hot brine at an average temperature of $80^{\circ}C$. The pulp was floated with 5 pounds of H_2SO_4 per ton, 2 pounds of $CuSO_4 \cdot 5 H_2O$, the emulsion, and 0.02 pound per ton of additional X-1. The total emulsion used for both stages equalled 0.25 pound of dixanthogen per ton and 0.04 pound of X-1.

(See table 18.)

TABLE 18. - Results of flotation test 195,
brine grind, acid and copper sulphate

Products	Weight, grams	Assay,		Distribution, percent	
		Ounces per ton Ag	Percent Pb	Ag	Pb
Heading.....	1,000	22.23	4.4	100.0	100.0
Sulphide concentrate...	184	95.05	7.0	78.7	29.2
Lead concentrate.....	75	42.35	25.5	14.2	43.3
Tailing.....	754	2.07	1.6	7.1	27.5

Satisfactory results were thus obtained with only 1-1/2 hours grinding, or three times the grinding period for conventional flotation tests.

After flotation, the tailing was panned to determine the loss of heavy metallics, as shown in table 19.

TABLE 19. - Results of panning, test 195-a

Products	Weight, grams	Assay		Distribution, percent	
		Ounces per ton Ag	Percent Pb	Ag	Pb
Panned from tailing...	20	2.85	2.0	0.3	0.9
Balance of tailing....	734	2.05	1.6	---	---

The concentrate panned from the tailing differed little in composition from the original tailing and contained less than 1 percent of the silver or of the lead. Apparently, the metallics were completely floated.

Heretofore, grinding in tap water has preceded the preliminary separation of the sulphides. If the pulp is to be ground next in saturated brine, it would be logical to use brine also for the preliminary grinding. Test 207 embodies this feature, the rest of the procedure being the same as for test 195. (See table 20.) With the dixanthogen emulsion, the end of the flotation stage was definite.

TABLE 20. - Results of flotation test 207,
brine grind

Products	Weight, grams	Assay		Distribution, percent	
		Ounces per ton Ag	Percent Pb	Ag	Pb
Heading.....	1,000	21.93	4.20	100.0	100.0
Pyrite concentrate...	197	95.10	6.50	85.4	30.5
Lead concentrate.....	50	39.90	42.00	9.1	50.2
Tailing.....	773	1.55	1.05	5.5	19.3

The use of brine instead of tap water in grinding prior to floating the sulphide portion has proved advantageous. The recovery of silver in the first concentrate was 85.4 percent and the calculated silver content of the tailing after this first stage was 3.88 ounces per ton. A comparison of this assay with the 5.7 ounces per ton reported in table 10 shows that the recovery of silver can be improved over that obtained by standard practice simply by grinding in brine instead of water and then using suitable flotation reagents.

The over-all results of this this test are the best obtained to date. The tailing is low in both lead and silver - 94.5 percent of the silver and 80.7 percent of the lead being recovered as concentrates, although the percentage of lead in the pyritic concentrate is unsatisfactorily high. Later, by grinding in brine in both stages and also depressing lead from the pyrite concentrate, 73.9 percent of the lead was recovered in the lead concentrate. To regulate conditions to those obtaining in actual practice, the concentration of brine in the flotation cell was adjusted to the same sodium chloride content, that obtained in both grinding stages. The recovery was not affected adversely; the tailing assayed 1.85 ounces of silver per ton and 0.9 percent of lead.

The foregoing gives the present status of the continuing work on minable ores. Some of the fundamentals of the process are discussed in the following section.

FUNDAMENTALS

Sponge Formation Using Specific Minerals

Procedure

Comparative tests were made to determine the extent to which specific oxidized minerals were reduced to metallic sponges when ground with iron balls. The general procedure follows: A charge of the mineral, usually 100 grams, was mixed with enough quartz to make 1,000 grams of synthetic ore. Six hundred ml of tap water was added to the charge in the ball mill. The charge was then ground for 4 hours in the closed mill. Next, the mill

was emptied and the sponge metal panned from the pulp. In some tests the pulp was thickened, ground for another 4 hours, and panned. The tailing from panning was sometimes floated to recover fine sponge. These metallic sponge products could have been separated by flotation instead of panning, and in many cases the separation would have been better. However, the object of this phase of the work was to investigate the formation of metallic sponge, and panning lessened contamination by unaltered mineral. Some of the oxidized minerals will float to some extent even when mild collectors are used.

Anglesite

The anglesite used in this test was argentiferous - the silver occurred largely as metallic inclusions.^{3/} After grinding, the products listed in table 21 were made:

TABLE 21. - Products from grinding anglesite

Products	Weight, grams	Assay, oz. per ton Ag	Assay, percent		Distribution, percent	
			Pb	Fe	Ag	Pb
Sponge A.....	68	40.0	46.5	7.10	55.3	47.4
Sponge B.....	53	21.6	38.2	13.20	23.3	30.4
Flotation concentrate.	22	30.5	44.2	2.65	14.2	15.2
Tailing.....	380	0.4	0.52	1.15	7.2	7.0
Heading calculated....	1,000	4.8	6.5	----	100.0	100.0

Sponge A - panned after 4 hours grinding.

Sponge B - panned after 4 hours additional grinding.

Lead	Sponge A	Sponge B	Flotation concentrate
1. Total Pb, percent.....	46.5	38.2	44.2
2. Total SO ₄ , percent.....	1.15	0.49	19.7
3. Pb equivalent of SO ₄ , percent.....	2.5	1.1	42.2
4. Metallic lead, (1) minus (3), percent	44.0	37.1	2.0
5. Metallic lead, percent of total Pb in product, (4) divided by (1).....	94.7	97.1	4.6
6. Metallic lead, percent of total lead in heading.....	44.9	29.9	0.7

Reference to these data brings out the following facts: Seventy-five percent of the anglesite was converted to metallic lead and recovered as a concentrate. In all, 93.0 percent of the lead and 92.8 percent of the silver were recovered at a ratio of concentration of 6.94 to 1. Much iron

^{3/} Head, R. E., Reference 7.

was abraded from the mill and balls. That all of this iron was not used to precipitate lead is shown by the assay for the residual iron in the sponges.

The large amount of unconsumed iron was believed to result from the excess of abrasive material in the charge. Accordingly, another grinding test was made, wherein the usual proportion of 100 grams of mineral to 900 grams of quartz was changed to 250 grams of anglesite and 750 grams of quartz. The results obtained when these two ratios are used follow in table 22:

TABLE 22. - Products from two different proportions of minerals ground

Ratio of anglesite to quartz	Product made	Assay, percent		Ratio Pb to Fe
		Pb	Fe	
100 grams anglesite to 900 grams quartz	(Panned concentrate 1	46.9	7.10	6.5
	(Panned concentrate 2	38.2	13.20	2.9
	(Flotation concentrate	44.2	2.50	16.7
250 grams anglesite to 750 grams quartz	(Panned concentrate 1	38.5	1.80	32.5
	(Panned concentrate 2	35.0	0.35	100.0
	(Flotation concentrate	12.0	0.25	48.0

The small amount of residual iron in the second test shows that the ratio of lead mineral to quartz during grinding may govern the percentage of the abraded iron that is utilized. It also appears that the arbitrary ratio commonly used in these tests, 100 grams of mineral to 900 grams of quartz, does not give an optimum mixture.

Cerussite

The cerussite and quartz mixture was ground in tap water, as was done with the anglesite. The reaction was slower than when anglesite was ground; however, 26.1 percent of the lead was panned out as metallic sponge. The cerussite was silver bearing, and 62.8 percent of the silver was recovered. The abraded iron remaining in the two sponges, A and B, was 25.2 percent and 21.0 percent, which was much higher than when anglesite was used. This is to be expected, as the amount of iron abraded was virtually the same, but less was precipitated; hence, less iron was consumed when the cerussite was ground. After grinding, an extremely pure variety of crystalline cerussite, sponge lead, was obtained as before. The variety of cerussite was unique, in that it contained only 0.176 ounce of silver per ton, and the silver in the tailing was too low to be determined.

Plain tap water (pH = 8.0) was used in all previous grinding tests; however, the effects of alkalinity induced by NaOH and Ca(OH)₂ have been determined. Five grams of NaOH was added to 600 ml of tap water; then the

cerussite-quartz mixture was ground as usual. The ground pulp was nearly white and upon panning and subsequently floating, no sponge lead was found. The presence of much unaltered abraded iron further verified the lack of sponge formation in this dilute solution of NaOH. Somewhat similar effects were noted when 4 grams of $\text{Ca}(\text{OH})_2$ was substituted for the NaOH. No sponge was found upon panning the ground pulp or was recovered by flotation in this alkaline circuit. Upon acidifying to a pH of 6.2, a small amount of sponge was floated.

The sponge formation during grinding was inhibited by the addition of alkalies; on the other hand, the substitution of 10 grams of acetic acid was helpful. Upon panning the pulp in acetic acid, twice as much lead was recovered as from the pulp ground in plain tap water. As usual, the recovery of silver was better than that of the lead.

Plumbojarosite

After grinding a mixture of plumbojarosite and quartz in tap water, sponge lead was panned from the pulp. The sponge was lower in grade and slightly less in amount than was obtained in the comparable test using cerussite. The plumbojarosite contained some silver, probably as argentojarosite, and the recovery of silver was poorer than that of the lead. The presence of some anglesite in the sample was suspected, so another test was made utilizing a carefully sorted sample of plumbojarosite. The amount of sponge lead recovered was somewhat less; however, sponge lead was definitely present in the pulp. From calculations based upon lead and sulphur analyses, 97.1 percent of the lead content of the first panned product was metallic.

Argentojarosite

Only 25 grams of argentojarosite was used, 25 grams of cerussite and 25 grams of anglesite being added to produce sponge lead to act as a collector. Enough quartz was added to make the usual 1,000-gram charge. The panned sponges were rich in silver; however, 69.6 percent of the silver and only 21.5 percent of the lead were lost in the tailing. Apparently, argentojarosite did not respond as well as cerussite and anglesite.

Cerargyrite

Only 10 grams of cerargyrite was used, as it assayed 3.070 ounces of Ag per ton. The rest of the mixture consisted of 15 grams of cerussite and 975 grams of quartz. After 4 hours grinding, 64.7 percent of the silver was collected with the metallics as a high-grade product. Cerargyrite responds well to metallization.

Malachite

There is a small amount of malachite and azurite in oxidized lead ores, and the degree to which these will metallize has bearing on the

problem. On grinding a mixture of malachite and quartz in tap water, and even in brine (not acidulated), the recovery of metallic copper by panning or flotation was less than with the lead minerals. The high percentages of residual abraded iron in the concentrates made (24.0 percent and 33.2 percent) also indicate the slow conversion of the malachite. Anglesite and malachite in equal amounts were substituted for the 100 grams of malachite used in the other tests. The panned sponge was many times richer in lead than in copper, and the recoveries of lead and silver, 89.3 and 93.4 percent, respectively, were higher than that of copper, 36.9 percent. It appears that the presence of both constituents promotes the mutual precipitation of these metals.

The first metallic sponges made from ores were formed by grinding in tap water, and the use of this medium in the subsequent study on synthetic ores followed naturally. The later work on ores has shown that much better results are obtained by grinding in brine; and a series of tests similar to the above, but employing brine instead of water, would doubtless show, in general, faster rates and better results for the various lead and silver minerals.

Nature of Gangue

A hard siliceous material was predominant in both ores and all synthetic mixtures used in these experiments. It is believed that the presence of some hard substance is necessary to maintain fresh rough surfaces on the iron balls and is essential to the process. No direct evidence is offered, but the following relevant facts are significant:

After grinding ocherous gold ores, little gold or silver was collected along with abraded iron; the addition of quartz before grinding was helpful. Also, when attempts were made to metallize and float lead and bismuth from a roasted metallurgical product rich in iron oxide, the recovery of these metals in the concentrate was poor. Actual tests on mixtures of lead minerals and soft gangue materials such as calcite, or on a calcareous oxidized lead ore, of course, will definitely establish the importance of abrasive material in the charge.

Mill Speed

The speed of the mill during grinding has some effect upon the physical condition of the sponge produced. As shown in tests 195-a and 196-A (tables 13 and 7), the tailings after flotation are virtually free of heavy lead. Although this condition obtains at standard mill speeds of 72 r.p.m., a heavy metallic product forms at slow speeds. If heavy lead pellets are formed they will not float. In test 177-A, a charge of 200 grams of anglesite and 800 grams of quartz was ground in water for 16 hours at the slow speed of 23 r.p.m. Upon scouring the mill and balls by grinding with water and pure quartz, 41.6 percent of the lead in the original charge was found to have been plated on the mill and balls.

At the high speed of 102 r.p.m., the balls centrifuged considerably,

little grinding was done, and very little sponge was formed. The same amounts of anglesite, water, balls, and the same mill were used as in the previous tests; apparently, the process of forming sponge lead is dependent upon the intimate contact between mineral and iron such as is brought about only by a true grinding action.

The standard speed, 72 r.p.m., used in all tests except those specifically cited, may not be the optimum, but at this speed sponge does form rapidly and is in proper physical condition to be recovered by flotation. The speed of 72 r.p.m. is nearly that recommended by the manufacturers for ordinary grinding.

Iron Consumption

The metallic sponges are produced at the expense of the metallic iron present, and the consumption of iron is expected to be high. Thus, theoretically, 0.27 pound of iron will be required to precipitate 1 pound of lead, and 5.4 pounds of iron will be required to convert 1 percent of lead per ton of ore. As there is some inefficiency, in that pieces of abraded iron may not be completely replaced by lead, it is estimated that about 7 pounds of iron would be consumed. However, part of the lead present in the ore can be floated without altering the minerals, and iron would not be consumed by them.

When pure quartz and pure lead minerals are ground together, the total iron in discharged products must equal that removed from the mill and balls. The following data were obtained from some of the earlier tests, where the grinding periods were longer and more iron was abraded: In test 149, 22.5 grams of iron was consumed and 50.1 grams of lead metallized, or 8.96 pounds of iron was consumed for each percent of lead metallized per ton of ore. In test 154, where there was less abrasive material and more available PbSO_4 , the iron consumed was only 13.04 grams and the total amount of lead recovered in both cases was about the same. When the rate of precipitation is accelerated, as by grinding in hot brine, the unconsumed abraded iron is small in amount and the iron consumption probably approaches nearer the theoretical. More work is needed on this phase of the problem.

Iron "Shot"

Several experiments were made with iron shot, a cheap source of iron that can be produced without cost of casting. These shot are minus 1/2-inch, and it was thought that they might be satisfactory for the secondary grinding steps in hot brine. It appeared that the additional surface per unit of weight, compared with the larger balls, might even prove to be a definite advantage, accelerating the rate of formation of sponge. Test 199, on the Tintic Standard ore, was similar to 195, except that shot was used instead of balls for grinding in hot brine. It was noted that the shot was partly covered with lead. Next, a similar experiment, 200, was made, using the Chief Consolidated ore, richer in lead. Shot again was used as the grinding media. The tailing was fairly satisfactory for this

type of test, assaying 2.6 percent Pb and 3.0 ounces of silver per ton of ore. However, the shot was heavily plated with metallic lead, and only 40 percent of the lead in the original charge was in the pulp discharged from the mill. Subsequently, most of the remaining 60 percent was scoured from the shot by grinding with water and quartz sand. The scourings, after most of the sand had been screened out, assayed 49.2 percent of lead. This retention of much of the lead by grinding media was disconcerting experimentally, as results obtained by batch grinding would have little significance. A continuous series of grinding tests will be required to determine (1) the extent to which the masking of the shot by the lead coating will inhibit the activity of the iron as a precipitant, and also (2) the physical condition of the metallic lead ultimately produced. Some pieces of coarse iron were found in the pulp; whether the source of this iron was largely prills removed from the new shot or the result of mechanical disintegration of the brittle shot would also be shown by continued use.

The literature on the treatment of oxidized ores often notes the high consumption of balls and the quantity of soluble iron salts in the ores. The iron in solution could readily come from reaction between a mineral such as anglesite and the balls during grinding, and the ore itself might contain little or no water-soluble salts.

Flotation of Sponge

The metallic sponges, clean abraded iron and free silver, appear to float well. The floatability of lead sponge is confirmed by the absence of lead in products panned from tailing, and the floatability of fresh metallic iron is shown by the fact that no abraded iron is found in the tailings on testing with an electromagnet. The increase in recovery of silver after long grinding indicates that the metallic silver particles (known to exist in the anglesite and cerussite) are probably freed when the lead minerals are decomposed and that the metallic silver itself is floatable. Also, when treating a metallurgical product containing "spangle" silver, a concentrate assaying 5,296 ounces per ton was floated.

In the earlier work, Pentasol and in some cases xanthates were used as collectors. Pine oil, Pentasol alcohol, B-23, and emulsol X-1 were used as frothers, pine oil less generally because other frothers, such as Pentasol alcohol and B-23, produced a more brittle froth with more sorting action in the bubble column.

A reagent combination that has proved satisfactory for floating metallics is a water emulsion of dixanthogen prepared with Emulsol X-1. As dixanthogen is relatively inert chemically and stable as to pH, it does not react with soluble salts and is not decomposed in an acidic circuit. Emulsol X-1, in addition to acting as an emulsifying agent dispersing the insoluble dixanthogen as microscopic droplets, also acts as a frother within a wide range of pH in solutions of widely differing composition. As the relative proportions of dixanthogen to X-1 greatly affect the frothing power of the reagent, it is customary to make the

emulsion slightly deficient in X-1 and then add more X-1 as required during flotation. Toward the end of the flotation operation, enough acid is usually added to make the pulp temporarily acid to methyl orange. The froth then rises rather slowly but is heavy with metallics; at the end, the froth becomes practically barren and is white.

Although the emulsions were usually prepared from amyl dixanthogen, the ethyl compound was successfully substituted in the last test on Tintic Standard ore, 0.30 pound per ton being used. Ethyl dixanthogen is prepared much cheaper and more simply than the amyl compound.

After so much grinding, the gangue is extremely fine and floats readily, and is brought up by too much collector. Even a slight excess of the frother, pine oil, will ordinarily prevent separation. It is believed that, when formed, the sponge occludes considerable fine gangue material; some apparently clean sponges will assay less than 50 percent of lead.

Noteworthy is the fact that anglesite, which is refractory to flotation either directly or by sulphidizing and floating, is adaptable to the metallizing process. Conversely, cerussite, which metallizes more slowly, responds well to sulphidizing and is probably the most easily floatable of the oxidized lead minerals. Anglesite plus cerussite contain most of the lead in ordinary oxidized ores.

SUMMARY

In attempting to apply the observation that abraided iron collected gold on long grinding in a ball mill to the treatment of refractory ores, an important discovery was made and its practical implications were investigated.

1. When siliceous, oxidized lead-silver ore containing such minerals as anglesite, cerussite, plumbogjarosite, argentojarosite, or cerargyrite, is ground in a ball mill driven at standard speed, a heavy sponge of metallized lead is found in the settled pulp and ferrous iron appears in the supernatant liquor. The sponge, abraided iron, and free silver may be floated with high recovery.

2. The reaction between the lead minerals and the iron balls of the mill results in the formation of these products.

3. These findings suggested a new method of attack on the long-standing problem of the treatment of oxidized lead-silver ores of Utah. Careful and detailed tests were made on such ores from two well-known mines in that State, where ore is being shipped to smelters without concentration, at present.

4. Chief Consolidated ore, almost completely oxidized, contained 57 percent silica, 8 percent lead, 11 ounces silver, and small amounts of other minerals, including gold. A series of grinding, flotation, and

panning tests was run with various reagents, including a sulphidizer. Grinding was done in brine and in acidic brine, cold and hot. Conditions were established by which 74 percent of the lead and 83 percent of the silver was recovered without sulphidizing, and further experimentation is expected to result in even higher recovery of valuable constituents.

5. Tintic Standard ore contained 68 percent silica, 4-1/2 percent lead, and 22 ounces silver, also minor amounts of other minerals, including gold. This ore is partly oxidized; however, it contains considerable pyrite and much of the silver is associated with the sulphides. In this case, the sulphides were floated before the ore was subjected to fine grinding. The tailings were ground in acidic brine, cold and hot, with and without a sulphidizer, and floated with various reagents. Eventually, 81 percent of the lead and 94 percent of the silver was recovered in a pyritic concentrate rather high in lead. The work is being continued, and adjustments may improve the metal recovery.

6. A reagent combination prepared with dixanthogen and Emulsol X-1 was found to be effective for flotation from a pulp containing large quantities of soluble salts. The reagent can be used over a wide range of pH, including acidic circuits.

7. The conditions, with regard to both the grinding and flotation steps are somewhat complicated, but good progress has been made in simplification, and it is expected that further developments will result in procedures that simulate standard treatment methods.

CONCLUSION

The discovery and development of the metallization process for the effective recovery of precious metals and lead from siliceous oxidized ores that have heretofore failed to give up their valuable constituents to a satisfactory extent by standard methods of beneficiation appears to offer an opportunity for improved practice at certain operating mills and may encourage the mining and milling of refractory ore that has not been susceptible to economic treatment in the past.

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REPORT OF INVESTIGATIONS

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IN THE UNITED STATES IN 1936



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INTRODUCTION

This paper presents the results of the second in the series of annual canvasses on the consumption of ferrous scrap and pig iron in the United States, inaugurated in 1935 by the Bureau of Mines and published in Report of Investigations 3329. The study was undertaken in response to requests from trade organizations and others for detailed statistics on this phase

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- ^{1/} The Bureau of Mines will welcome reprinting of this paper, provided the following footnote acknowledgment is used: "Reprinted from Bureau of Mines Report of Investigations 3366."
- ^{2/} Mineral economist, Metal Economics Division, Bureau of Mines.
- ^{3/} Assistant mineral economist, Metal Economics Division, Bureau of Mines.
- ^{4/} Metal Economics Division, Bureau of Mines.

of the iron and steel industry. Thus, for the first time, producers, buyers, sellers, and consumers of iron and steel scrap have had an accurate statistical picture of this important raw material. As originally contemplated, the study was to be continued on an annual basis, but, owing to budgetary limitations for the fiscal year 1937-38, the Bureau may be unable to conduct the canvass for 1937.

The present report is confined to presentation of factual material and important changes as revealed by the 2-year record. No attempt has been made to analyze the statistics in the light of the recent controversy over the limitation of scrap exports, but it is believed that students of this problem will find much essential basic data in the various tables presented herein.

The Bureau gratefully acknowledges the generous cooperation of the many concerns that have contributed to the survey, and sincere thanks are extended to the thousands of auditors and accountants throughout the country who have compiled painstakingly the many details requested.

GENERAL SUMMARY

The total consumption of ferrous scrap and pig iron in 1936 increased 41 percent over 1935. Of the 66,456,767 gross tons used, home scrap accounted for 28.4 percent, purchased scrap 26.3 percent, and pig iron 45.3 percent. In 1935 total consumption amounted to 47,035,793 tons, of which 28.4 percent was home scrap, 27.8 percent purchased scrap, and 43.8 percent pig iron. Thus, in 1936, as compared with 1935, relatively more pig iron and less purchased scrap were used, the total quantity of pig iron consumed having increased 46 percent, whereas that of purchased scrap increased only 34 percent. This trend undoubtedly reflected the higher prices for scrap in 1936, but the record does not indicate that this fact necessitated major adjustments in furnace operations in any section of the country.

Consumption of ferrous raw materials in the iron and steel industries of the Southwestern, Pacific Coast, and New England States, which depend largely on scrap, compared favorably with the average for the entire country, the average percentage increase in the three regions being 37, as compared with 41 for the entire country. The relative use of purchased scrap in these outlying areas actually increased from 57 percent of the total consumption of scrap and pig iron in 1935 to 60 percent in 1936, but they consumed less than 7 percent of the total purchased scrap used in 1936. In the six largest steel-producing States, Pennsylvania, Ohio, Indiana, Illinois, Michigan, and New York, which in 1936 consumed 78 percent of the total scrap and 83 percent of the pig iron, the proportion of purchased scrap to total scrap and pig iron decreased from 26 percent in 1935 to 24 percent in 1936, largely due to preferential use of pig iron in open-hearth furnaces in these States.

Higher scrap prices apparently had only slight effects in furnace operations in 1936. In the open hearths, which account for nearly three-fourths of the total consumption of ferrous scrap and pig iron, the ratio of purchased scrap to total charge declined from 28.3 percent in 1935 to 26.0 percent in

1936. A contributing factor to this decline was the increased output of duplex steel. This practice, however, was not so pronounced in 1936 as in previous years of comparable scrap prices. In cupola furnaces, which include a large number of relatively small operators, the use of purchased scrap increased more than that of pig iron.

The importance of scrap from the viewpoint of conservation is illustrated by the relative amounts of scrap and ore used in the domestic iron and steel industry. In 1936 the total scrap consumed was equivalent to 137 percent of the iron content of all domestic and foreign iron and manganiferous iron ores used, and purchased scrap alone equaled 66 percent of the iron content of the ores. In 1935 the respective percentages were 146 and 72.

SCOPE OF REPORT

In its canvass of the consumers of ferrous scrap and pig iron, the Bureau of Mines obtains data only on that portion of the scrap used in remelting processes. Thus, material that is salvaged and reused in its original condition or is reshaped, without melting, into other serviceable articles is omitted. In order to simplify the canvass, no details are sought regarding the ordinary trade classifications of the scrap consumed and no attempt is made to obtain data on its value or cost at consumers plants.

Statistics have been compiled showing consumption of scrap and pig iron by States and by types of furnaces. In keeping with the traditional policy of the Bureau not to disclose, except by permission, details of individual operations, it has been necessary in some instances to combine figures for some States. All such combinations have been made with a view to revealing details of consumption by types of furnaces rather than by geographic subdivisions.

From the standpoint of its position in the industrial cycle, scrap may be divided into several categories. It may be termed "old" or "new" scrap, according to whether it represents material salvaged from articles discarded after use, such as old automobiles, stoves, and the usual assortment of junk, or whether it is a byproduct of manufacturing operations, such as clippings and trimmings. Another division may be made on the basis of origin. Thus, "home" or "plant" scrap refers to material originating within the same plant in which it is consumed, whereas, "purchased" scrap refers to materials originating elsewhere and purchased by the consumer.

In this study no attempt has been made to differentiate between old and new scrap for the reason that the large volume of iron and steel accumulated by waste-material dealers contains both old and new material that cannot be segregated statistically. However, consumers were asked to report their use of home and purchased scrap separately. Home or plant scrap, as used herein, refers to scrap produced at the plant of the establishment reporting and includes new material, such as runners, spills, risers, skulls, butts, cropping clippings, borings, turnings, mill and hammer scale, cinder, defective products discarded, etc., as well as old material. Although some establishments

may utilize high proportions of old plant scrap for a limited period; in general the quantities of material included under this heading largely represent recycled or run-around scrap, which may be considered by many as merely an increment in working stocks rather than actual consumption. However, this scrap constitutes a definite part of the furnace charges and must be included in order to present a complete picture of raw material consumption in steel manufacture. This is particularly true as long as ingot production (which includes a large part of this recycled scrap) remains the yardstick for measuring activity in the steel industry.

The term "purchased scrap" includes purchased material as well as scrap transferred from other plants under the same control and scrap received under exchange contracts or conversion agreements. The proportion of transferred, exchange, or conversion scrap included in the purchased scrap item is doubtless quite low, most of the tonnage having been obtained by purchase from scrap dealers, brokers, or scrap producers.

Coverage.— Although reports were not received from all consumers, it is believed that the final tabulations include plants that consume about 99 percent of the total ferrous scrap and pig iron in uses that involve remelting of the scrap. Estimated coverages by different types of consuming equipment are as follows:

<u>Consuming equipment</u>	<u>Percent coverage</u>
Open-hearth furnaces.....	100
Bessemer converters.....	100
Electric-steel furnaces.....	97
Cupola and air furnaces.....	95
Blast furnaces.....	100
Weighted average.....	99

Lists of consumers, including iron and steel mills, ferrous foundries, and blast furnaces, were assembled from several sources, and about 3,700 questionnaires were sent out. Of these, about 3,300 were returned from 2,900 active plants and 400 inactive plants. Of the 400 plants for which reports were not received, many were inactive and many were small foundries whose aggregate capacity represents a very small percentage of the total.

Most of the reports received were based on actual records, although many small companies, chiefly operators of foundries, stated that certain of their figures were only reasonable estimates. Numerous concerns, particularly those using cupolas, did not report consumption of home or plant (recycled) scrap. It was necessary, therefore, to estimate this item in several instances. Thus, the consumption of home or plant scrap in cupola furnaces, as reported herein for 1936, contains an estimate of 397,000 gross tons, based on the assumption that the recycled scrap represented 30 percent of the cupola charge. The total consumption of home scrap in all types of furnaces includes an estimate of only 507,000 tons, or 2.7 percent of the total.

CONSUMPTION OF FERROUS SCRAP AND PIG IRON, BY DISTRICTS AND STATES

Ferrous scrap or pig iron are consumed in all of the 48 States, in the District of Columbia, and in Alaska (see table 1). The great concentration of consumption, however, is in the steel-making centers of the North Central and Middle Atlantic States. These areas include the six largest consuming States, which used 78 percent of the total scrap, 83 percent of the pig iron, and 80 percent of the total scrap and pig iron charged to furnaces in 1936. These States and the percentage of the total ferrous scrap and pig iron each consumed in 1936 were as follows: Pennsylvania 27, Ohio 22, Indiana 11, Illinois 9, Michigan 6, and New York 5. Alabama, Maryland, and West Virginia ranked 7th, 8th, and 9th, respectively, and Kentucky ranked 10th. No other State consumed as much as 1,000,000 tons of scrap and pig iron in 1936. Nine of the 10 principal consuming States showed increases in consumption of ferrous raw materials in 1936 ranging from 8 percent in Kentucky to 60 percent in Pennsylvania. West Virginia's consumption declined about 1 percent, all of which was due to a decrease in pig-iron supply, the use of scrap having increased about 1 percent.

Table 2 shows the total consumption of scrap and pig iron by geographic subdivisions and the percentage of the total contributed by home scrap, purchased scrap, and pig iron. It will be noted that scrap constitutes by far the greater part of the raw materials used in the Southwestern, Pacific Coast, and New England States. These regions, however, accounted for less than 5 percent of the total scrap consumed in 1936. In all areas except the Southwestern and Pacific Coast States proportionately less scrap was used in 1936 than in 1935, although the declines in most instances were relatively small. Apparently, higher prices for scrap in 1936 did not result in major adjustments in furnace charges in any part of the country.

CONSUMPTION OF FERROUS SCRAP AND PIG IRON, BY TYPE OF FURNACE

The relative position of the open-hearth furnace as the chief consumer of ferrous scrap and pig iron is shown in table 3. More than double the amount of ferrous raw materials was used in this type of furnace in 1936 than in all other types combined. In 1936 the open-hearth furnace took 72 percent of the total ferrous scrap consumed (72 percent in 1935), 73 percent of the home scrap (72 percent in 1935), 72 percent of the purchased scrap (73 percent in 1935), and 73 percent of the pig iron (71 percent in 1935).

More than 90 percent of the domestic steel output, as measured in steel ingots and castings, is made in open-hearth furnaces, charges to which in 1936 consisted of 54 percent ferrous scrap and 46 percent pig iron, as compared with 57 and 43 percent, respectively, in 1935. Of the total scrap used in open hearths in 1936, 48 percent was purchased scrap, as compared with 50 percent in 1935.

TABLE 1.-Total consumption of ferrous scrap and pig iron in the United States in 1936, by districts and States

District and State	Number of active plants report- ing	Scrap						Pig iron	
		Home		Purchased		Total		Gross tons	Per- cent of total
		Gross tons	Per- cent of total	Gross tons	Per- cent of total	Gross tons	Per- cent of total		
NEW ENGLAND:									
Connecticut.....	64	59,494	0.32	133,783	0.77	193,277	0.53	79,208	0.27
Maine.....	21	9,241	.05	9,284	.05	18,525	.05	9,257	.03
New Hampshire.....	17								
Massachusetts.....	106	83,641	.44	207,991	1.19	291,632	.80	75,389	.25
Rhode Island.....	15	21,311	.11	34,549	.20	55,860	.16	25,983	.09
Vermont.....	15	3,618	.02	3,708	.02	7,326	.02	3,866	.01
Total, 1936....	238	177,305	.94	389,315	2.23	566,620	1.56	193,703	.65
Total, 1935....	232	144,408	1.08	305,221	2.33	449,629	1.70	146,656	.71
MIDDLE ATLANTIC:									
Delaware.....	9	183,814	.97	452,317	2.59	636,131	1.75	215,460	.71
New Jersey.....	94								
New York.....	237	867,363	4.59	813,054	4.66	1,680,417	4.62	1,371,661	4.56
Pennsylvania.....	464	4,714,527	24.94	3,834,558	21.97	8,549,085	23.52	9,074,405	30.15
Total, 1936....	804	5,765,704	30.50	5,099,929	29.22	10,865,633	29.89	10,661,526	35.42
Total, 1935....	770	3,803,287	28.50	3,201,118	24.49	7,004,405	26.52	6,445,123	31.26
SOUTHEASTERN:									
Alabama.....	77	643,596	3.41	534,509	3.06	1,178,105	3.24	1,453,524	4.83
District of Columbia.....	4	5,973	.03	13		5,986	.02	501	
Florida.....	17	29,291	.15	83,248	.48	112,539	.31	41,051	.14
Georgia.....	46								
Kentucky.....	21	907,208	4.80	670,902	3.84	1,578,110	4.34	1,489,375	4.95
Maryland.....	31								
West Virginia.....	32	353,453	1.87	605,357	3.47	958,810	2.64	648,882	2.15
Mississippi.....	11	842	.07	1,506	.01	2,348	.08	351	.04
North Carolina....	37	11,601		17,340	.10	28,941		11,064	
South Carolina....	18	1,340		2,507	.01	3,847		1,912	
Tennessee.....	51	103,215	.55	111,120	.64	214,335	.59	142,994	.48
Virginia.....	63								
Total, 1936....	408	2,056,519	10.88	2,026,502	11.61	4,083,021	11.23	3,789,654	12.59
Total, 1935....	370	1,567,671	11.74	1,748,596	13.38	3,316,267	12.55	2,865,364	13.90
SOUTHWESTERN:									
Arkansas.....	13	13,825	.07	56,279	.32	70,104	.19	2,273	.02
Oklahoma.....	19								
Louisiana.....	22								
Texas.....	50	21,501	.12	59,010	.34	80,511	.22	4,699	
Total, 1936....	104	35,326	.19	115,289	.66	150,615	.41	6,972	.02
Total, 1935....	98	20,922	.16	75,348	.58	96,270	.37	5,010	.02

TABLE 1.-Total consumption of ferrous scrap and pig iron in the
United States in 1936, by districts and States - Continued

District and State	Number of active plants report- ing	Scrap						Pig iron	
		Home		Purchased		Total		Gross tons	Per- cent of total
		Gross tons	Per- cent of total	Gross tons	Per- cent of total	Gross tons	Per- cent of total		
NORTH CENTRAL:									
Illinois.....	217	1,744,705	9.23	1,624,370	9.31	3,369,075	9.27	2,770,746	9.21
Indiana.....	128	2,209,821	11.69	1,668,389	9.56	3,878,210	10.67	3,473,415	11.54
Iowa.....	51	52,938	.28	66,762	.38	119,700	.33	62,576	.21
Minnesota.....	67	101,888	.54	227,725	1.30	329,613	.91	46,024	.15
Missouri.....	62	54,361	.29	401,930	2.30	456,291	1.25	40,367	.13
Kansas.....	36	14,186	.08	43,926	.25	58,112	.16	3,726	.01
Nebraska.....	15								
Michigan.....	187	1,801,941	9.53	1,392,372	7.98	3,194,313	8.78	1,567,890	5.21
Wisconsin.....	130								
North Dakota.....	2	144		170		314		9	
South Dakota.....	1								
Ohio.....	334	4,464,449	23.62	3,448,475	19.75	7,912,924	21.76	7,013,146	23.30
Total, 1936....	1,230	10,444,433	55.26	8,874,119	50.83	19,318,552	53.13	14,977,899	49.76
Total, 1935....	1,144	7,490,057	56.12	7,161,041	54.80	14,651,098	55.46	10,875,718	52.74
ROCKY MOUNTAIN:									
Arizona.....	8	7,616	.04	11,488	.07	19,104	.05	72	1.07
Nevada.....	4								
New Mexico.....	1	155,327	.82	239,862	1.37	395,189	1.09	320,514	
Colorado.....	24								
Utah.....	15	39	.02	200	.03	239	.03	1	.01
Idaho.....	1								
Wyoming.....	2	3,880		5,766		9,646		2,804	
Montana.....	7								
Total, 1936....	62	166,862	.88	257,316	1.47	424,178	1.17	323,391	1.08
Total, 1935....	58	109,796	.82	125,259	.96	235,055	.89	174,507	.85
PACIFIC COAST:									
Alaska.....	1	35,227	.19	139,117	.80	174,344	.48	8,223	.03
Oregon.....	26								
Washington.....	60	220,013	1.16	555,157	3.18	775,170	2.13	137,266	.45
California.....	130								
Total, 1936....	217	255,240	1.35	694,274	3.98	949,514	2.61	145,489	.48
Total, 1935....	193	210,611	1.58	451,995	3.46	662,606	2.51	108,085	.52
UNITED STATES									
TOTAL, 1936....	1/3,063	18,901,389	100.00	17,456,744	100.00	36,358,133	100.00	30,098,634	100.00
1935....	1/2,865	13,346,752	100.00	13,068,578	100.00	26,415,330	100.00	20,620,463	100.00

1/ Where 2 or more separate departments, such as blast-furnace department, open-hearth department, foundry department, etc., are located at the same place and are operated by 1 establishment, each of these departments appears as a plant in the total figure.

TABLE 2.-- Total scrap and pig iron consumed and percentage of total consumption contributed by home scrap, purchased scrap, and pig iron in 1935 and 1936, by districts.

District	1935				1936					
	Total scrap and pig iron used gross tons	Percent of total used			Total scrap and pig iron used gross tons	Percent of total used				
		Scrap		Pig iron		Scrap		Pig iron		
		Home	Purchased			Total	Home		Purchased	Total
New England.....	596,285	24.2	51.2	75.4	24.6	760,323	23.3	51.2	74.5	25.5
Middle Atlantic.	13,449,528	28.3	23.8	52.1	47.9	21,527,159	26.8	23.7	50.5	49.5
South Eastern...	6,181,631	25.4	28.3	53.7	46.3	7,872,675	26.1	25.8	51.9	48.1
South Western...	101,280	20.7	74.4	95.1	4.9	157,587	22.4	73.2	95.6	4.4
North Central...	25,526,816	29.3	28.1	57.4	42.6	34,296,451	30.4	25.9	56.3	43.7
Rocky Mountain..	409,562	26.8	30.6	57.4	42.6	747,569	22.3	34.4	56.7	43.3
Pacific Coast...	770,691	27.3	58.7	86.0	14.0	1,095,003	23.3	63.4	86.7	13.3
Total.....	47,035,793	28.4	27.8	56.2	43.8	66,456,767	28.4	26.3	54.7	45.3

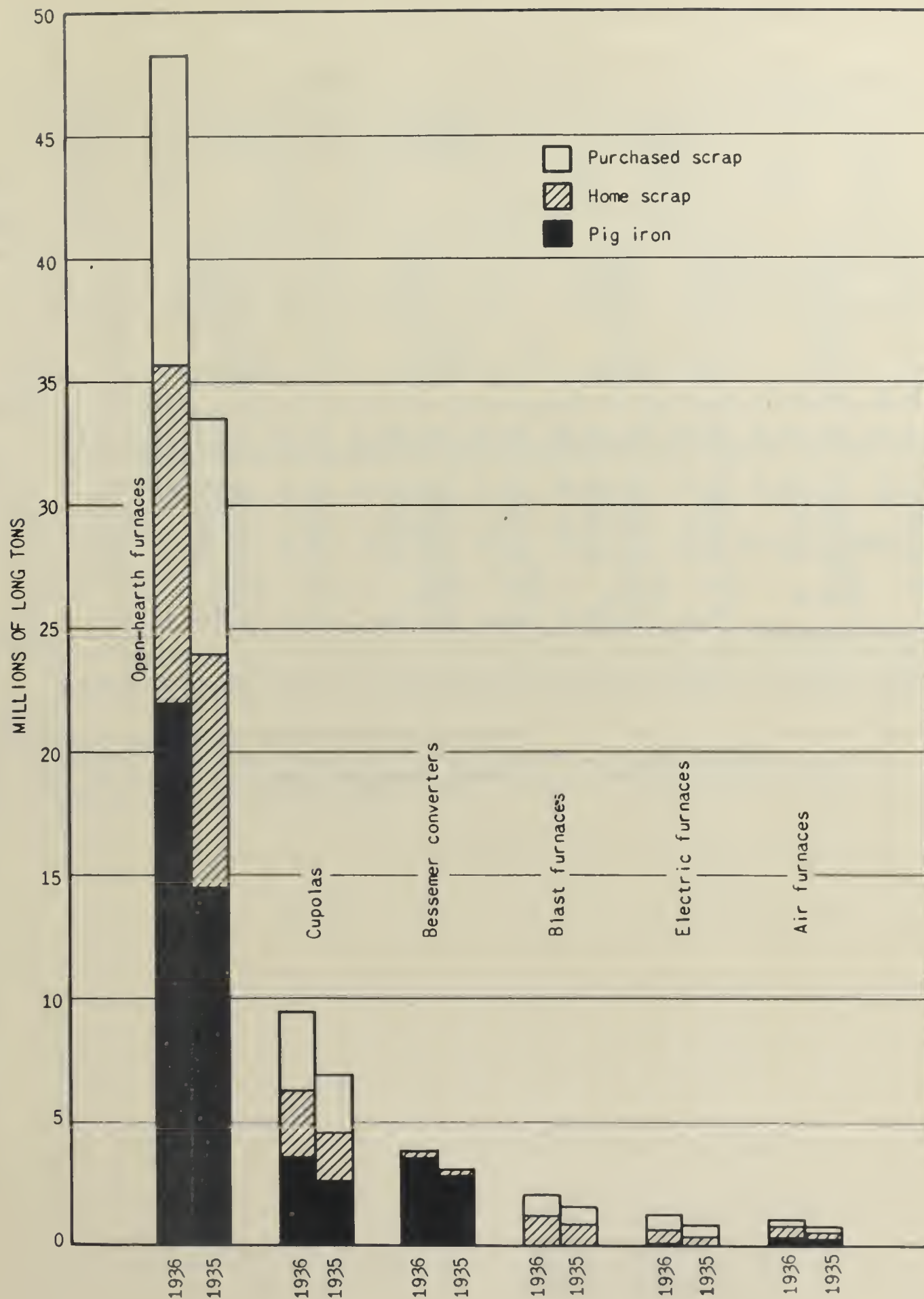


Figure 1.— Consumption of ferrous scrap and pig iron in the United States, 1935 and 1936, by types of furnace.

TABLE 3.- Consumption of ferrous scrap and pig iron in the United States in 1935-36, by type of furnace

Type of furnace or equipment	Number of active plants reporting	Scrap						Pig iron	
		Home		Purchased		Total		Gross tons	Per-cent of total
		Gross tons	Per-cent of total	Gross tons	Per-cent of total	Gross tons	Per-cent of total		
1935									
Open-hearth.....	127	9,589,017	71.85	9,530,610	72.93	19,119,627	72.38	1/14,575,239	1/70.68
Bessemer.....	30	212,862	1.60	6,452	.05	219,314	.83	1/2,911,719	1/14.12
Electric.....	217	464,783	3.48	450,776	3.45	915,559	3.47	33,186	.16
Cupola.....	2,287	1,916,835	14.36	2,241,788	17.15	4,158,623	15.74	2/2,675,827	2/12.98
Air.....	115	278,140	2.08	168,103	1.29	446,243	1.69	295,008	1.43
Crucible.....	10	244	.01	609	.03	853	.02	566	.07
Puddling.....	5	1,371		4,020		5,391		13,492	
Blast.....	67	883,500	6.62	666,220	5.10	1,549,720	5.87
Direct castings..	7	2/115,426	2/.56
	3/2,865	13,346,752	100.00	13,068,578	100.00	26,415,330	100.00	20,620,463	100.00
1936									
Open-hearth.....	136	13,748,882	72.74	12,546,809	71.87	26,295,691	72.32	21,960,842	72.96
Bessemer.....	29	226,724	1.20	12,632	.07	239,356	.66	3,635,562	12.08
Electric.....	240	641,451	3.39	605,978	3.47	1,247,429	3.43	22,866	.08
Cupola.....	2,436	2,656,843	14.06	3,157,590	18.09	5,814,433	15.99	2/3,633,720	2/12.07
Air.....	116	441,353	2.33	223,154	1.28	664,507	1.83	407,038	1.35
Crucible.....	13	369	.02	372	.04	741	.03	34	.10
Puddling.....	6	2,767		6,899		9,666		30,498	
Blast.....	77	1,183,000	6.26	903,310	5.18	2,086,310	5.74
Direct castings..	10	2/408,074	2/1.36
	3/3,063	18,901,389	100.00	17,456,744	100.00	36,358,133	100.00	30,098,634	100.00

1/Revised figures.

2/Some pig iron used in making direct castings included in cupola.

3/Where 2 or more separate departments, such as blast-furnace department, open-hearth department, foundry department, etc., are located at the same place and are operated by 1 establishment, each of these departments appears as a plant in the total figure.

Cupolas were the second largest consumers of scrap, taking 18 percent of the total purchased scrap used in 1936 (17 percent in 1935), 14 percent of the home scrap (14 percent in 1935), and 16 percent of the total scrap (16 percent in 1935). In pig-iron consumption, cupolas and Bessemer converters each took about equal quantities (12 percent of the total) in 1936, while in 1935 Bessemer furnaces accounted for 14 percent of the total and cupolas 13 percent. Bessemer converters consume only minor quantities of scrap.

The percentage composition of charges to the various types of furnaces in 1935 and 1936, in terms of scrap and pig iron, is shown in table 4. It will be noted that the proportions of total scrap and pig iron used, as well as the relative amounts of purchased and home scrap, were changed only slightly in 1936, as compared with 1935. Apparently, higher prices for scrap in 1936 did not influence furnace operation to any great extent, although open-hearth operators appear to have favored pig iron in their charges.

TABLE 4.-- Proportion of purchased and home scrap and pig iron used in furnace charges, 1935-36, in percent

Type of furnace	1935				1936			
	Scrap			Pig iron	Scrap			Pig iron
	Purchased	Home	Total		Purchased	Home	Total	
Open-hearth..	1/ 28.3	1/ 28.4	1/ 56.7	1/ 43.3	26.0	28.5	54.5	45.5
Bessemer.....	1/ .2	1/ 6.8	1/ 7.0	1/ 93.0	.3	5.9	6.2	93.8
Electric.....	47.5	49.0	96.5	3.5	47.7	50.5	98.2	1.8
Cupola.....	32.8	28.1	60.9	39.1	33.4	28.1	61.5	38.5
Air.....	22.7	37.5	60.2	39.8	20.8	41.2	62.0	38.0
Crucible.....	42.9	17.2	60.1	39.9	48.0	47.6	95.6	4.4
Puddling.....	21.3	7.3	28.6	71.4	17.2	6.9	24.1	75.9
Blast.....	43.0	57.0	100.0	—	43.3	56.7	100.0	—

1/ Revised figures

Open-hearth furnaces. (Table 5.)— The total quantity of ferrous scrap and pig iron consumed in domestic open hearths, as reported by 136 operating plants, was 48,256,533 gross tons, an increase of 43 percent over the 33,694,866 tons consumed in 1935. Of the 1936 total, purchased scrap accounted for 26.0 percent, home scrap 28.5 percent, and pig iron 45.5 percent; in 1935 the percentages were 28.3, 28.4, and 43.3, respectively. Thus, the use of pig iron increased 51 percent, whereas scrap consumption increased only 38 percent.

A factor contributing to the greater use of pig iron in 1936 was the large increase in the use of duplex steel, so-called "synthetic scrap", prompted by higher prices for scrap. According to the American Iron and Steel Institute, production of duplex steel increased from 960,020 tons in 1935 to 2,099,102 tons in 1936. It should be pointed out, however, that duplexing operations have fluctuated with the price of scrap for many years and that the production of duplex in 1936 was not abnormally high. From 1923 to 1936 the ratio of duplex to basic open-hearth steel production has ranged from 8.6 to 1.9 percent

TABLE 5.- Consumption of ferrous scrap and pig iron in open-hearth furnaces
in the United States in 1936, by districts and States

District and State	Number of active plants report- ing	Scrap						Pig iron	
		Home		Purchased		Total		Gross tons	Per- cent of total
		Gross tons	Per- cent of total	Gross tons	Per- cent of total	Gross tons	Per- cent of total		
NEW ENGLAND:									
Connecticut.....	1	53.661	0.39	231.591	1.84	285.252	1.08	41.038	0.19
Massachusetts.....	1								
Rhode Island.....	1								
Total, 1936.....	3	53.661	0.39	231.591	1.84	285.252	1.08	41.038	0.19
Total, 1935.....	3	50.910	.53	188.378	1.98	239.288	1.25	30.475	1/ .21
MIDDLE ATLANTIC:									
Delaware.....	1	733.651	5.34	633.597	5.05	1,367.248	5.20	1,206.614	5.49
New Jersey.....	2								
New York.....	7								
Pennsylvania.....	51	3,964.617	28.83	3,160.397	25.19	7,125.014	27.10	7,251.119	33.02
Total, 1936.....	61	4,698.268	34.17	3,793.994	30.24	8,492.262	32.30	8,457.733	38.51
Total, 1935.....	56	2,989.123	31.17	2,344.799	24.60	5,333.922	27.90	1/4,848.151	1/ 33.26
SOUTHEASTERN AND SOUTH- WESTERN:									
Alabama.....	3	350.254	2.55	454.379	3.62	804.633	3.06	993.107	4.52
Georgia.....	1								
Oklahoma.....	1								
District of Columbia....	1	1.391	.01	1,391	253
Kentucky.....	2	1,103.083	8.02	1,126.188	8.98	2,229.271	8.48	1,753.021	7.99
Maryland.....	1								
West Virginia.....	2								
Total, 1936.....	11	1,454.728	10.58	1,580.567	12.60	3,035.295	11.54	2,746.381	12.51
Total, 1935.....	11	1,151.864	12.01	1,368.201	14.35	2,520.065	13.18	2,149.560	1/14.75
NORTH-CENTRAL:									
Illinois.....	11	1,236.539	8.99	1,170.115	9.33	2,406.654	9.15	1,682.322	7.66
Indiana.....	7	2,000.340	14.55	1,446.661	11.53	3,447.001	13.11	3,141.247	14.30
Michigan.....	4	463.106	3.37	523.481	4.17	986.587	3.75	729.931	3.32
Iowa.....	1	18.912	.14	317.009	2.53	335.921	1.28	10.998	.05
Missouri.....	2								
Minnesota.....	1								
Wisconsin.....	2	94.821	.69	173.593	1.38	268.414	1.02	30.025	.14
Ohio.....	26	3,434.872	24.98	2,627.665	20.94	6,062.537	23.06	4,740.508	21.59
Total, 1936.....	54	7,248.590	52.72	6,258.524	49.88	13,507.114	51.37	10,335.031	47.06
Total, 1935.....	50	5,192.068	54.15	5,203.553	54.60	10,395.621	54.37	7,322.864	1/50.24
PACIFIC COAST AND ROCKY MOUNTAIN:									
California.....	5	293.635	2.14	682.133	5.44	975.768	3.71	380.659	1.73
Colorado.....	1								
Washington.....	1								
Total, 1936.....	7	293.635	2.14	682.133	5.44	975.768	3.71	380.659	1.73
Total, 1935.....	7	205.052	2.14	425.679	4.47	630.731	3.30	224.189	1/1.54
UNITED STATES TOTAL, 1936.	136	13,748.882	100.00	12,546.809	100.00	26,295.691	100.00	21,960.842	100.00
UNITED STATES TOTAL, 1935.	127	9,589.017	100.00	9,530.610	100.00	19,119.627	100.00	1/14,575.239	100.00

1/ Revised figures.

and corresponded roughly with the respective high and low prices for scrap during that period. In 1936, the ratio was 4.9 percent, which compares with 5.9 percent in 1927, 5.2 percent in 1928, and 6.0 percent in 1930, years in which the average prices of scrap were approximately the same as in 1936.

Higher proportions of purchased scrap are used in open-hearth operations in areas remote from pig-iron producing centers. For example, in 1936 purchased scrap represented 71 percent of the charge in New England and 50 percent in the Pacific Coast area, while in Pennsylvania and Ohio the proportions were 22 percent and 24 percent, respectively. The practice of using scrap exclusively is relatively rare. In 1936, only four plants out of a total of 136 operated on a 100-percent scrap basis; they consumed only 503,367 tons, slightly more than 1 percent of the total consumption of ferrous raw materials in open hearths.

According to the Annual Statistical Report of the American Iron and Steel Institute for 1936, the total amount of open-hearth steel (basic plus acid) produced in the United States in 1936 was 43,536,128 gross tons. Since this is equivalent to over 90 percent of the scrap and pig iron consumed in open-hearth furnaces as reported to the Bureau of Mines, a loss of over 9 percent in this process is indicated, most of which is accounted for by the elimination of impurities present in the pig iron. The indicated loss in 1935 was slightly less than 9 percent, but a larger loss was to be expected in 1936 because of the higher proportion of pig iron used.

Pennsylvania, the leading steel producer, led all States in 1936 in the consumption of ferrous scrap and pig iron in the open hearth and was followed by Ohio, Indiana, and Illinois.

Bessemer converters. (Table 6.)- The proportion of scrap consumed in Bessemer converters is very low, amounting to only 6 percent of the total charge in 1936, and virtually all of this consisted of home or plant scrap. All of the very small tonnage of purchased scrap consumed in converters was used in small steel-foundry plants.

The consumption of ferrous scrap and pig iron in Bessemer converters in 1936 was 3,874,918 gross tons, an increase of 24 percent over 1935. The American Iron and Steel Institute reported a production of 3,458,457 tons of Bessemer steel in 1936, indicating that in American practice there is a loss of about 10 percent in manufacturing steel by this method, due to burning out of impurities, oxidation, spillage, etc.

Electric steel furnaces. (Table 7.)- A total of 1,270,295 gross tons of scrap and pig iron was consumed in electric furnaces in 1936, an increase of 34 percent over 1935. Scrap comprises the bulk of the ferrous raw material fed to this type of equipment, pig iron accounting for less than 2 percent of the total used in 1936. Of the 240 active plants reporting, 99 operated exclusively on scrap and consumed 307,491 tons, or about 24 percent of the total scrap and pig iron used in electric furnaces.

Ohio led all States in scrap consumption in the electric furnace, followed by Pennsylvania, Michigan, Illinois, and New York.

TABLE 6.- Consumption of ferrous scrap and pig iron in Bessemer converters
in the United States in 1936, by districts and States

District and State	Number of active plants report- ing	Scrap						Pig iron	
		Home		Purchased		Total		Gross tons	Per- cent of total
		Gross tons	Per- cent of total	Gross tons	Per- cent of total	Gross tons	Per- cent of total		
NEW ENGLAND AND MIDDLE ATLANTIC:									
Delaware.....	1	561	0.25	641	5.07	1,202	0.50	658	0.02
Massachusetts.....	1								
New Jersey.....	1								
New York.....	1								
Pennsylvania.....	8	65,148	28.73	1,243	9.84	66,391	27.74	1,041,226	28.64
Total, 1936.....	12	65,709	28.98	1,884	14.91	67,593	28.24	1,041,884	28.66
Total, 1935.....	13	54,457	25.58	1,446	22.41	55,903	25.49	1,817,331	1/28.07
SOUTHEASTERN AND SOUTH- WESTERN:									
Maryland.....	1	23,656	10.43	1,855	14.69	25,511	10.66	247,218	6.80
West Virginia.....	1								
Louisiana.....	2								
Texas.....	2								
Total, 1936.....	6	23,656	10.43	1,855	14.69	25,511	10.66	247,218	6.80
Total, 1935.....	4	15,344	7.21	130	2.02	15,474	7.06	174,335	1/5.99
NORTH-CENTRAL:									
Illinois.....	2	21,123	9.32	4,469	35.38	25,592	10.69	627,757	17.27
Indiana.....	1								
Michigan.....	1								
Missouri.....	1								
Ohio.....	6	116,236	51.27	4,424	35.02	120,660	50.41	1,718,703	47.27
Total, 1936.....	11	137,359	60.59	8,893	70.40	146,252	61.10	2,346,460	64.54
Total, 1935.....	13	143,061	67.21	4,876	75.57	147,937	67.45	1,920,053	1/65.94
UNITED STATES TOTAL, 1936.									
UNITED STATES TOTAL, 1935.	29	226,724	100.00	12,632	100.00	239,356	100.00	3,635,562	100.00
	30	212,862	100.00	6,452	100.00	219,314	100.00	1,291,719	100.00

1/Revised figures.

TABLE 7.-Consumption of ferrous scrap and pig iron in electric-steel furnaces in the United States in 1936, by districts and States

District and State	Number of active plants report- ing	Scrap						Pig iron	
		Home		Purchased		Total		Gross tons	Per- cent of total
		Gross tons	Per- cent of total	Gross tons	Per- cent of total	Gross tons	Per- cent of total		
NEW ENGLAND:									
Connecticut.....	6	6,550	1.02	6,095	1.01	12,645	1.02	944	4.13
New Hampshire.....	1								
Rhode Island.....	1								
Massachusetts.....	6	4,212	.66	7,036	1.16	11,248	.90	129	.56
Total, 1936.....	14	10,762	1.68	13,131	2.17	23,893	1.92	1,073	4.69
Total, 1935.....	13	10,631	2.29	6,603	1.46	17,234	1.88	1,424	4.29
MIDDLE ATLANTIC:									
Delaware.....	1	7,375	1.15	21,576	3.56	28,951	2.32	545	2.38
New Jersey.....	6								
New York.....	16	35,558	5.54	53,630	8.85	89,188	7.15	2,535	11.09
Pennsylvania.....	42	131,462	20.50	117,592	19.41	249,054	19.97	6,466	28.28
Total, 1936.....	65	174,395	27.19	192,798	31.82	367,193	29.44	9,546	41.75
Total, 1935.....	59	114,984	24.74	147,486	32.72	262,470	28.67	6,428	19.37
SOUTHEASTERN:									
District of Columbia..	1	4,362	.68	-----	-----	4,362	.35	8	.04
Maryland.....	1	2,287	.36	10,616	1.75	12,903	1.03	14	.06
West Virginia.....	2								
Alabama.....	2	4,690	.73	10,688	1.76	15,378	1.23	105	.46
Florida.....	1								
Tennessee.....	1								
Virginia.....	3	4,691	.73	-----	-----	4,691	.38	37	.16
Total, 1936.....	11	16,030	2.50	21,304	3.51	37,334	2.99	164	.72
Total, 1935.....	13	13,324	2.87	19,717	4.37	33,041	3.61	438	1.32
SOUTHWESTERN:									
Arkansas.....	1	11,975	1.86	11,514	1.90	23,489	1.88	106	.46
Oklahoma.....	1								
Texas.....	4								
Louisiana.....	3	312	.05	1,365	.23	1,677	.14	-----	-----
Total, 1936.....	9	12,287	1.91	12,879	2.13	25,166	2.02	106	.46
Total, 1935.....	9	6,166	1.33	7,512	1.67	13,678	1.49	214	.64
NORTH-CENTRAL:									
Illinois.....	14	76,065	11.86	36,656	6.05	112,721	9.04	2,272	9.94
Indiana.....	7	12,542	1.95	22,020	3.63	34,562	2.77	616	2.69
Iowa.....	2	5,652	.88	9,466	1.56	15,118	1.21	98	.43
Kansas.....	1								
Nebraska.....	1								
Michigan.....	24	44,390	6.92	74,251	12.25	118,641	9.51	2,473	10.81
Minnesota.....	3	1,654	.26	3,511	.58	5,165	.41	55	.24
Missouri.....	7	4,407	.69	5,141	.85	9,548	.77	546	2.39
Ohio.....	25	218,340	34.04	119,187	19.67	337,527	27.06	3,550	15.53
Wisconsin.....	14	30,217	4.71	39,578	6.53	69,795	5.59	1,783	7.80
Total, 1936.....	98	393,267	61.31	309,810	51.12	703,077	56.36	11,393	49.83
Total, 1935.....	83	295,286	63.53	237,966	52.79	533,262	58.24	23,766	71.62

TABLE 7.-Consumption of ferrous scrap and pig iron in electric-steel furnaces
in the United States in 1936, by districts and States - Continued

District and State	Number of active plants report- ing	Scrap						Pig iron	
		Home		Purchased		Total		Gross tons	Per- cent of total
		Gross tons	Per- cent of total	Gross tons	Per- cent of total	Gross tons	Per- cent of total		
ROCKY MOUNTAIN:									
Arizona.....	2	4,091	.64	4,729	.78	8,820	.70	-----	-----
Colorado.....	2								
Utah.....	1								
Total, 1936.....	5	4,091	.64	4,729	.78	8,820	.70	-----	-----
Total, 1935.....	6	4,676	1.00	3,247	.72	7,923	.87	-----	-----
PACIFIC COAST:									
Alaska.....	1	2,031	.32	5,852	.97	7,783	.63	9	.04
Oregon.....	3								
California.....	21	22,885	3.56	23,595	3.89	46,480	3.73	446	1.95
Washington.....	13	5,703	.89	21,880	3.61	27,583	2.21	129	.56
Total, 1936.....	38	30,619	4.77	51,327	8.47	81,946	6.57	584	2.55
Total, 1935.....	34	19,716	4.24	28,245	6.27	47,961	5.24	916	2.76
UNITED STATES TOTAL, 1936									
UNITED STATES TOTAL, 1935	240	641,451	100.00	605,978	100.00	1,247,429	100.00	22,866	100.00
	217	464,783	100.00	450,776	100.00	915,559	100.00	33,186	100.00

Cupolas. (Table 8.)- Total consumption of ferrous scrap and pig iron in cupola furnaces in 1936 was 9,448,153 gross tons, an increase of 38 percent over 1935. Use of purchased scrap increased 41 percent over 1935, home scrap 39 percent, total scrap 40 percent, and pig iron 36 percent. The larger demand for raw materials in foundries, which include a large number of small consumers, thus was met by relatively larger quantities of purchased scrap, notwithstanding the higher prices for that commodity in 1936. The practice of operating cupolas on 100 percent scrap is not at all uncommon, as 369 plants reported consumption of 594,996 tons of ferrous scrap without the use of pig iron in 1936.

The figures for cupola furnaces in 1936 are based on returns from 2,436 operators and represent an estimated coverage of 95 percent of the total consumption in this type of equipment. Many of the plants included in the above figure are small, and some do not keep complete records, particularly on recycled scrap. For this reason it has been necessary to estimate the home scrap item for some plants. Therefore, the total consumption of 2,656,843 tons of home scrap shown for 1936 includes 397,000 tons that have been estimated on the basis of practice at other plants engaged in the same or similar industries.

Michigan was the principal consumer of scrap in the cupola, followed by Illinois, Ohio, Pennsylvania, and New York.

Air furnaces. (Table 9.)- Air furnaces consumed 1,071,545 gross tons of ferrous scrap and pig iron in 1936, an increase of 45 percent over 1935. The use of purchased scrap increased 32 percent, home scrap 59 percent, total scrap 49 percent, and pig iron 38 percent. For this type of equipment there was, thus, a relatively greater use of pig iron than purchased scrap in 1936. Only three operators of air furnaces reported exclusive use of scrap in 1936, the quantity consumed amounting to only 810 tons.

Michigan led in scrap consumption in air furnaces, followed, in order, by Ohio, Illinois, Pennsylvania, and Indiana.

Blast furnaces. (Table 10.)- Scrap accounts for only a small proportion of the metal-bearing materials consumed in blast furnaces. The other materials used in 1936 were 51,835,257 gross tons of iron and mangani ferous iron ores, 3,521,544 tons of cinder and scale, and 1,340,600 tons of flue dust. Total consumption of scrap in 1936, as reported by 77 plants operating blast furnaces, was 2,086,310 tons, an increase of 35 percent over 1935. Of the 1936 total, 903,310 tons (43 percent) was purchased scrap and 1,183,000 tons (57 percent) home scrap. These tonnages represent increases of 36 percent and 34 percent, respectively, over the preceding year.

The proportion of scrap used in blast furnaces declined in 1936. In that year the total scrap used was equivalent to 6.9 percent of the pig iron produced, as compared with 7.4 percent in 1935. Purchased scrap used was equivalent to 3.0 percent of the pig iron produced in 1936 and 3.2 percent in 1935.

The blast furnaces in Ohio consumed more scrap than those in any other State.

TABLE 8.- Consumption of ferrous scrap and pig iron in cupola furnaces
in the United States in 1936, by districts and States

District and State	Number of active plants report- ing	Scrap						Pig iron	
		Home		Purchased		Total		Gross tons	Per- cent of total
		Gross tons	Per- cent of total	Gross tons	Per- cent of total	Gross tons	Per- cent of total		
NEW ENGLAND:									
Connecticut.....	50	38,374	1.44	35,778	1.13	74,152	1.27	46,042	1.27
Maine.....	20	5,571	.21	6,965	.22	12,536	.22	7,288	.20
Massachusetts.....	93	36,464	1.37	74,999	2.38	111,463	1.92	53,848	1.48
New Hampshire.....	14	1,533	.06	2,024	.06	3,557	.06	1,471	.04
Rhode Island.....	12	13,025	.49	10,766	.34	23,791	.41	15,556	.43
Vermont.....	15	3,618	.14	3,708	.12	7,326	.13	3,866	.10
Total, 1936.....	204	98,585	3.71	134,240	4.25	232,825	4.01	128,071	3.52
Total, 1935.....	199	73,074	3.81	103,274	4.61	176,348	4.24	99,254	3.71
MIDDLE ATLANTIC:									
Delaware.....	5	2,102	.08	3,942	.12	6,044	.10	3,367	.09
New Jersey.....	80	69,510	2.62	205,686	6.52	275,196	4.73	151,142	4.45
New York.....	194	129,383	4.87	239,000	7.57	368,383	6.34	177,149	4.88
Pennsylvania.....	307	200,533	7.55	329,680	10.44	530,213	9.12	501,028	13.79
Total, 1936.....	586	401,528	15.12	778,308	24.65	1,179,836	20.29	842,686	23.19
Total, 1935.....	568	310,448	16.20	501,982	22.39	812,430	19.53	592,946	22.16
SOUTHEASTERN:									
Alabama.....	66	138,262	5.20	112,580	3.57	250,842	4.31	479,316	13.19
District of Columbia....	2	24,416	.92	38,712	1.23	63,128	1.09	35,808	.99
Maryland.....	27								
Florida.....	16	1,129	.04	2,224	.07	3,353	.06	329	.01
Georgia.....	45	11,373	.43	18,646	.59	30,019	.51	21,821	.60
Kentucky.....	17	17,156	.65	11,486	.36	28,642	.49	48,889	1.35
Mississippi.....	11	842	.03	1,506	.05	2,348	.04	351	.01
North Carolina.....	37	11,601	.44	17,340	.55	28,941	.50	11,064	.30
South Carolina.....	18	1,340	.05	2,507	.08	3,847	.07	1,912	.05
Tennessee.....	49	69,563	2.62	43,622	1.38	113,185	1.95	122,851	3.38
Virginia.....	59	26,950	1.01	64,550	2.04	91,500	1.57	20,003	.55
West Virginia.....	23	10,716	.40	14,420	.45	25,136	.43	47,151	1.30
Total, 1936.....	370	313,348	11.79	327,593	10.37	640,941	11.02	789,495	21.73
Total, 1935.....	330	215,775	11.26	245,816	10.97	461,591	11.10	537,202	20.08
SOUTHWESTERN:									
Arkansas.....	12	775	.02	4,390	.14	5,165	.09	220	.03
Louisiana.....	17	3,641	.14	12,448	.40	16,089	.28	937	
Oklahoma.....	17	2,787	.11	8,315	.26	11,102	.19	926	
Texas.....	43	11,821	.45	52,067	1.65	63,888	1.10	4,380	.12
Total, 1936.....	89	19,024	.72	77,220	2.45	96,244	1.66	6,463	.18
Total, 1935.....	86	14,601	.76	65,989	2.94	80,590	1.94	4,709	.17

TABLE 8.- Consumption of ferrous scrap and pig iron in cupola furnaces
in the United States in 1936, by districts and States - Continued

District and State	Number of active plants report- ing	Scrap						Pig iron	
		Home		Purchased		Total		Gross tons	Per- cent of total
		Gross tons	Per- cent of total	Gross tons	Per- cent of total	Gross tons	Per- cent of total		
NORTH-CENTRAL:									
Illinois.....	169	248,565	9.36	331,067	10.49	579,632	9.97	470,569	12.95
Indiana.....	101	106,975	4.03	168,223	5.33	275,198	4.73	113,302	3.12
Iowa.....	46	46,067	1.73	51,621	1.63	97,688	1.68	56,269	1.55
Kansas.....	34	8,414	.32	32,724	1.04	41,138	.71	2,167	.06
Michigan.....	144	937,354	35.28	474,142	15.02	1,411,496	24.28	592,571	16.31
Minnesota.....	60	16,305	.61	55,543	1.76	71,848	1.24	20,249	.56
Missouri.....	51	28,525	1.07	84,038	2.66	112,563	1.93	27,976	.77
Nebraska.....	14	2,616	.10	6,299	.20	8,915	.16	1,507	.04
North Dakota.....	2	144		170		314		9	
South Dakota.....	1								
Ohio.....	237	227,156	8.55	306,750	9.71	533,906	9.18	356,061	9.80
Wisconsin.....	104	118,500	4.46	121,225	3.84	239,725	4.12	140,162	3.85
Total, 1936.....	963	1,740,621	65.51	1,631,802	51.68	3,372,423	58.00	1,780,842	49.01
Total, 1935.....	905	1,218,567	63.57	1,204,918	53.75	2,423,485	58.28	1,385,246	51.77
ROCKY MOUNTAIN:									
Arizona.....	6	3,396	.13	10,256	.32	13,652	.23	11	.43
Colorado.....	19	10,033	.38	29,834	.95	39,867	.68	15,762	
Idaho.....	1	99	.15	465	.20	564	.01	46	
New Mexico.....	1								
Wyoming.....	2	3,880	.14	5,766	.03	9,646	.17	2,804	.08
Montana.....	7								
Nevada.....	4	3,762	.14	942	.03	4,704	.08	16	
Utah.....	13	9,064	.34	20,918	.66	29,982	.52	18,606	.51
Total, 1936.....	53	30,234	1.14	68,181	2.16	98,415	1.69	37,245	1.02
Total, 1935.....	48	20,833	1.09	52,696	2.35	73,529	1.77	24,274	.91
PACIFIC COAST:									
California.....	102	45,471	1.71	118,164	3.74	163,635	2.82	42,212	1.16
Oregon.....	23	2,171	.08	4,513	.14	6,684	.11	3,497	.10
Washington.....	46	5,861	.22	17,569	.56	23,430	.40	3,209	.09
Total, 1936.....	171	53,503	2.01	140,246	4.44	193,749	3.33	48,918	1.35
Total, 1935.....	151	63,537	3.31	67,113	2.99	130,650	3.14	32,196	1.20
UNITED STATES TOTAL, 1936.....	2,436	2,656,843	100.00	3,157,590	100.00	5,814,433	100.00	1/3,633,720	100.00
UNITED STATES TOTAL, 1935.....	2,287	1,916,835	100.00	2,241,788	100.00	4,158,623	100.00	1/2,675,827	100.00

1/Includes some pig iron used in making direct castings.

TABLE 9.-Consumption of ferrous scrap and pig iron in air furnaces
in the United States in 1936, by districts and States

District and State	Number of active plants report- ing	Scrap						Pig iron	
		Home		Purchased		Total		Gross tons	Per- cent of total
		Gross tons	Per- cent of total	Gross tons	Per- cent of total	Gross tons	Per- cent of total		
NEW ENGLAND:									
Connecticut.....	6	6,076	1.38	4,434	1.99	10,510	1.58	11,725	2.88
Massachusetts.....	3	5,958	1.35	4,688	2.10	10,646	1.60	9,687	2.38
Maine.....	1	374	.08	714	.32	1,088	.17	1,899	.47
New Hampshire.....	1								
Rhode Island.....	1								
Total, 1936.....	12	12,408	2.81	9,836	4.41	22,244	3.35	23,311	5.73
Total, 1935.....	13	9,653	3.47	6,663	3.96	16,316	3.66	15,391	5.22
MIDDLE ATLANTIC:									
Delaware.....	1	8,972	2.03	2,315	1.04	11,287	1.70	7,030	1.73
New Jersey.....	3								
New York.....	9	22,216	5.04	9,109	4.08	31,325	4.71	22,788	5.60
Pennsylvania.....	22	44,496	10.08	38,303	17.16	82,799	12.46	64,610	15.87
Total, 1936.....	35	75,684	17.15	49,727	22.28	125,411	18.87	94,428	23.20
Total, 1935.....	35	55,396	19.92	36,828	21.91	92,224	20.67	59,882	20.30
SOUTHEASTERN AND SOUTHWESTERN:									
West Virginia.....	2	6,310	1.43	1,242	.56	7,552	1.14	4,199	1.03
Texas.....	1								
Total, 1936.....	3	6,310	1.43	1,242	.56	7,552	1.14	4,199	1.03
Total, 1935.....	4	3,812	1.37	6,420	3.82	10,232	2.29	3,916	1.33
NORTH CENTRAL:									
Illinois.....	15	71,367	16.17	41,912	18.78	113,279	17.05	73,903	18.16
Indiana.....	9	39,865	9.03	29,934	13.41	69,799	10.50	42,844	10.53
Michigan.....	9	114,858	26.02	20,143	9.03	135,001	20.32	71,568	17.58
Iowa.....	1	6,927	1.57	3,332	1.49	10,259	1.54	9,516	2.34
Minnesota.....	1								
Missouri.....	1								
Ohio.....	17	77,918	17.66	35,567	15.94	113,485	17.08	63,146	15.51
Wisconsin.....	10	35,493	8.04	30,886	13.84	66,379	9.99	22,649	5.56
Total, 1936.....	63	346,428	78.49	161,774	72.49	508,202	76.48	283,626	69.68
Total, 1935.....	60	208,543	74.98	117,918	70.15	326,461	73.16	214,802	72.81
ROCKY MOUNTAIN AND PACIFIC COAST:									
Colorado.....	1	523	.12	575	.26	1,098	.16	1,474	.36
California.....	2								
Total, 1936.....	3	523	.12	575	.26	1,098	.16	1,474	.36
Total, 1935.....	3	736	.26	274	.16	1,010	.22	1,017	.34
UNITED STATES TOTAL, 1936	116	441,353	100.00	223,154	100.00	664,507	100.00	407,038	100.00
UNITED STATES TOTAL, 1935	115	278,140	100.00	168,103	100.00	446,243	100.00	295,008	100.00

**TABLE 10.-Consumption of ferrous scrap in blast furnaces in the
United States in 1936, by districts and States**

District and State	Number of active plants report- ing	Home		Purchased		Total	
		Gross tons	Per- cent of total	Gross tons	Per- cent of total	Gross tons	Per- cent of total
NEW ENGLAND AND MIDDLE ATLANTIC:							
Massachusetts.....	1	43,408	3.67	96,028	10.63	139,436	6.68
New York.....	7						
Pennsylvania.....	22	306,667	25.92	183,245	20.28	489,912	23.48
Total, 1936.....	30	350,075	29.59	279,273	30.91	629,348	30.16
Total, 1935.....	27	277,411	31.40	164,267	24.66	441,678	28.50
SOUTHEASTERN:							
Alabama.....	6	172,348	14.57	45,259	5.01	217,607	10.43
Kentucky.....	1	114	.01	56,083	6.21	56,197	2.69
Maryland.....	1	52,830	4.46	6,410	.71	59,240	2.84
Tennessee.....	1	342	.03	137	.02	479	.02
West Virginia.....	2	19,628	1.66	8,435	.93	28,063	1.35
Total, 1936.....	11	245,262	20.73	116,324	12.88	361,586	17.33
Total, 1935.....	10	167,700	18.98	110,159	16.53	277,859	17.93
NORTH-CENTRAL:							
Illinois.....	5	110,537	9.35	42,046	4.65	152,583	7.31
Indiana.....	3	30,748	2.60	1,551	.17	32,299	1.55
Iowa.....	1	-----	-----	4,767	.53	4,767	.23
Michigan.....	5	44,519	3.76	96,740	10.71	141,259	6.77
Minnesota.....	2	2,436	.21	3,330	.37	5,766	.28
Ohio.....	18	389,926	32.96	354,880	39.29	744,806	35.70
Total, 1936.....	34	578,166	48.88	503,314	55.72	1,081,480	51.84
Total, 1935.....	28	432,532	48.96	391,794	58.81	824,326	53.19
ROCKY MOUNTAIN:							
Colorado.....	1	9,497	.80	4,399	.49	13,896	.67
Utah.....	1						
Total, 1936.....	2	9,497	.80	4,399	.49	13,896	.67
Total, 1935.....	2	5,857	.66	-----	-----	5,857	.38
UNITED STATES TOTAL, 1936.							
UNITED STATES TOTAL, 1935.	77	1,183,000	100.00	903,510	100.00	2,086,510	100.00
	67	883,500	100.00	666,220	100.00	1,549,720	100.00

Crucible and puddling furnaces. (Table 11.)— Only minor quantities of ferrous raw materials are used in crucible and puddling furnaces, whose combined output of iron and steel is very small.

TABLE 11.— Consumption of ferrous scrap and pig iron in crucible and puddling furnaces in the United States in 1936,
by districts and States

District and State	Number of active plants report- ing	Scrap						Pig iron	
		Home		Purchased		Total		Gross tons	Per- cent of total
		Gross tons	Per- cent of total	Gross tons	Per- cent of total	Gross tons	Per- cent of total		
NEW ENGLAND:									
Connecticut....	1	117	3.73	339	4.66	456	4.38	—	—
Massachusetts..	1								
New Hampshire..	1								
Total, 1936...	3	117	3.73	339	4.66	456	4.38	—	—
Total, 1935...	3	20	1.24	273	5.90	293	4.69	20	0.14
MIDDLE ATLANTIC:									
New Jersey.....	2	213	6.79	25	.35	238	2.29	5,054	16.55
New York.....	2								
Pennsylvania...	7	1,604	51.15	4,098	56.36	5,702	54.79	22,872	74.91
Total, 1936...	11	1,817	57.94	4,123	56.71	5,940	57.08	27,926	91.46
Total, 1935...	9	1,538	98.33	4,340	93.76	5,928	94.94	14,038	99.86
SOUTHEASTERN AND NORTH-CENTRAL:									
Kansas.....	1	1,202	38.33	2,809	38.63	4,011	38.54	2,606	8.54
Kentucky.....	1								
Ohio.....	2								
Virginia.....	1								
Total, 1936...	5	1,202	38.33	2,809	38.63	4,011	38.54	2,606	8.54
Total, 1935...	3	7	.43	16	.34	23	.37	—	—
UNITED STATES									
TOTAL, 1936.....	19	3,136	100.00	7,271	100.00	10,407	100.00	30,532	100.00
UNITED STATES									
TOTAL, 1935.....	15	1,615	100.00	4,629	100.00	6,244	100.00	14,058	100.00

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